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DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

GOLD DREDGING IN THE UNITED STATES

BY

CHARLES JANIN

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GOLD DREDGING IN THE UNITED STATES.

By CHARLES JANIN.

INTRODUCTION.

The recovery of gold from sands and gravels is one of the oldest forms of mining; it antedates history and has been practiced by savage peoples. In North America the search for placer gold has been a powerful agency in the exploration and development of unknown regions and has profoundly influenced the course of our civilization. With the progress of invention, the improvements in machinery, and the availability of large capital for the conduct of great operations, the tendency in placer, as in many other branches of mining, has been toward the working of extensive but comparatively low-grade deposits. The early miner washed the rich spots in stream beds or terraces with his pan, his long tom, and his primitive sluice. Then followed the growth of hydraulic mining and the use of giants supplied with water under pressures of hundreds of pounds to the square inch that could work at a profit gravels whose gold tenor was less than 3 cents to the cubic yard. Finally came the dredge that could handle deposits lying so far below water level as to be beyond attack by other methods.

Although the gold dredge was first successful on a commercial scale in New Zealand, it has reached its present strength and efficiency in this country and its development is a monument to the daring, perseverance and technical skill of the many men, mine owners and engineers who have each contributed something to the final result. The Bureau of Mines, authorized to conduct investigations for the purpose of increasing efficiency in the mineral industries, had its attention called some years ago to the need of a comprehensive report on gold dredging in the United States. The director of the bureau recognized that such a report should summarize the development of the gold dredge, should describe in detail the essential features of present-day dredges, should discuss the facts that determine whether a placer deposit can be profitably dredged, and should point out the approved methods of prospecting placer ground and of operating dredges.

At the request of the director, the author undertook the preparation of such a report and has been intermittently engaged on the task for several years. This bulletin is the outcome of his labors. He hopes that it will prove of interest to designers and builders of dredges, to mining engineers, and to persons who may contemplate investing capital in placer-mining ventures.

ACKNOWLEDGMENTS.

In the preparation of this report the writer has obtained information from many sources. He takes pleasure in saying that nearly every request for information was met with courteous response and hearty cooperation. Although it would be difficult to mention the names of all who have given information at different times, the writer feels especially indebted to the following men: W. P. Hammon, Emery Oliver, Charles Gardiner, George Carr, M. L. Summers, E. G. Thunen, Charles Coil, Louis Hopfield, Frank Griffin, Hennen Jennings, Charles Kammerer, W. H. James, C. G. Leeson, E. J. Hopkins, and Charles Haley.

Of the construction companies, the Bucyrus Co., through its engineer, Mr. Rosenblatt; the Marion Steam Shovel Co., through Mr. Felsmuth; the New York Engineering Co., through Mr. Ludlum and Mr. Loweric; the Union Construction Co., through Messrs. Johnson and Peake, the Union Iron Works, through Mr. Hearst, and the Yuba Construction Co., have supplied considerable information. Many of the photographs used have been furnished by these companies.

The Edgar Allen American Manganese Co. and the Taylor Wharton Iron & Steel Co. have also been generous in giving information.

The chapter on dredge construction was written almost entirely by George E. Sibbett. This chapter is especially mentioned, as the author of this bulletin considers it by far the best presentation of the subject ever published.

HISTORY.

Little is to be gained by narrating in detail the history of gold dredging, but a report of this kind should contain some mention of the progress and development of the industry. For this reason a brief historical summary is given.

Successful gold dredging in the United States is a matter of not more than 20 years' standing. Efforts were made even before 1890 to dredge the beds of rivers, but it was not until 1895 that success began to crown the efforts of the pioneers in this country. The first dredges to work in the United States were erected in California; and a machine was constructed even as early as 1849 for the purpose of scooping the gravel from the inaccessible beds of the Sacramento River. This machine was sent around the Horn from New York, but soon after its arrival it sank to the bottom of the river, where no doubt it still remains. During succeeding years many attempts at gold dredging were made in California and elsewhere in the United States.

EARLY DREDGES.

It is curious that, in spite of knowledge that might have been obtained from New Zealand, many of the "dredges" first tried were of the suction type which, so far as the writer has been able to discover, has never proved a success in gold dredging.

SPOON DREDGE.

An early, if not the earliest, type of dredge was the so-called "spoon dredge." In 1865 spoon dredges worked by hand had been tried with some success in New Zealand on rich spots on the Clutha River. A photograph of the first dredge of this kind is reproduced in Plate I, *A*. Plate II, *A*, shows a model of a dredge used by natives in the Philippine Islands. In the California dredges steam was used for power and the spoons were made of iron and weighed upward of a thousand pounds, whereas the spoon of the early New Zealand dredge consisted of an iron ring or cutting edge to which was fastened a rawhide bag. These spoons were placed at the end of a long pole and dragged along the river bottom by means of a rope attached to a windlass. When the bags were filled they were lifted to the barge by means of the windlass and their contents dumped into sluices or washed in ordinary rockers by hand.

One California "spoon" dredge in 1893 is reported to have recovered $40 a day for a short period. The New Zealand boats, hand operated, recovered as much as 80 ounces a week of six working days; but those with steam power were reported a failure.

The orange-peel bucket excavators, such as are used on harbor work, were tried in various places and were used in California in 1886;[a] two were also tried at Ruby, Mont. However, excavations of this type were uniformly unsuccessful. The steam shovel has been a partial success in gold dredging. With modifications and an auxilary washing plant, and placed either on a floating pontoon or used as a dry land excavator, with a long or short boom, the steam shovel has been tried in many fields. The first one recorded in gold dredging was used in New Zealand in 1870.

The gold dredge with bucket elevators was designed by a man named Scott and put in operation in New Zealand in 1882.[b] This dredge at first used steam power, but as the boilers were worn out and the engines were too large and heavy for the pontoons, the consumption of fuel proved excessive for the capacity of the dredge. Consequently the engines and boilers were discarded. Side paddle wheels placed on the pontoon and driven by the current of the river were used to elevate the dredging buckets. Sheet-iron buckets attached to the paddle wheels lifted the water for washing the material dug. This dredge (Pl. I, *B*), after a varied career, finally proved a success, and at one time more than 12 dredges of this type were busy on the river. One current-wheel dredge, according to the report of the Minister of the Mines, New Zealand, recovered £3,750 during two months operation.

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[a] California State Mining Bureau, Eighth ann. rept., State Mineralogist, 1887-88, pp. 100-102.

[b] Cutten, W. H., New Zealand Inst. Min. Eng., Aug. 23, 1898.

The advantages of the current-driven bucket dredge were: Small cost, no fuel, few hands required. The disadvantages were: Dependence on the current for power, and hence necessity of keeping well out in the stream where the efficiency was regulated by the depth of the water; during a considerable part of the year the water may have been too deep or too shallow to permit work.

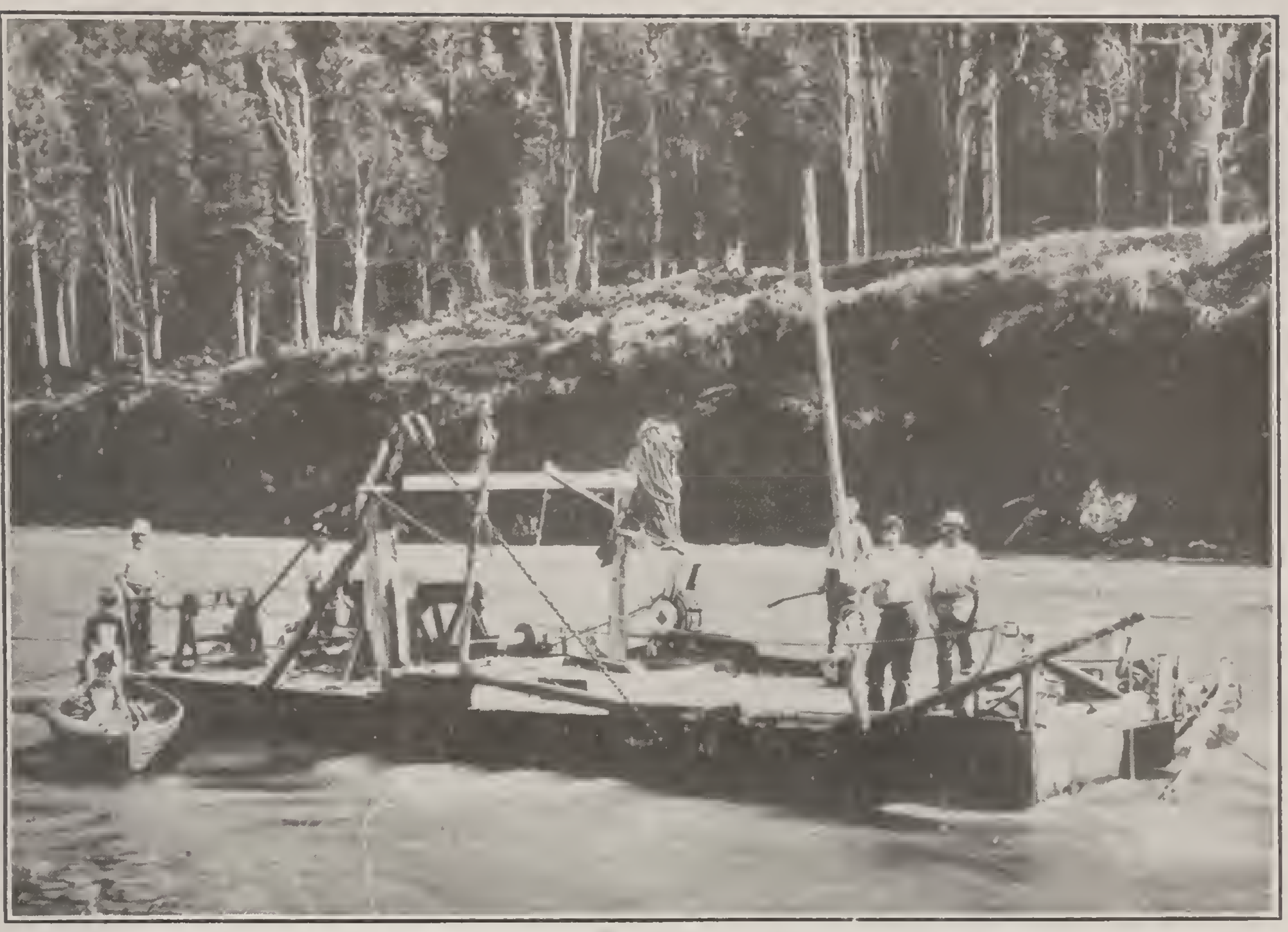
STEAM DREDGES.

The first steam gold dredge was designed by Charles McQueen and was in operation on the Clutha River in 1882. This dredge was equipped with a double line of elevator buckets, one on each side of the hull, which was of iron, 70 feet long, 15 feet beam, and 6 feet deep. The buckets discharged into two revolving screens. The fine dirt went into the sluice boxes, which at first were equipped with revolving copper pans to amalgamate the gold. The pans were soon discarded as useless and the gold was caught on cocoa matting. The two ladders were not wholly a success, as dredging first on one side and then the other occasionally rolled the decks under water; then the strong current of the river at times made the boat almost unmanageable. During 16 years of operation, first with two ladders and afterwards with one, the dredge is credited with having recovered nearly £60,000 ($300,000). The dredge is shown in Plate III.

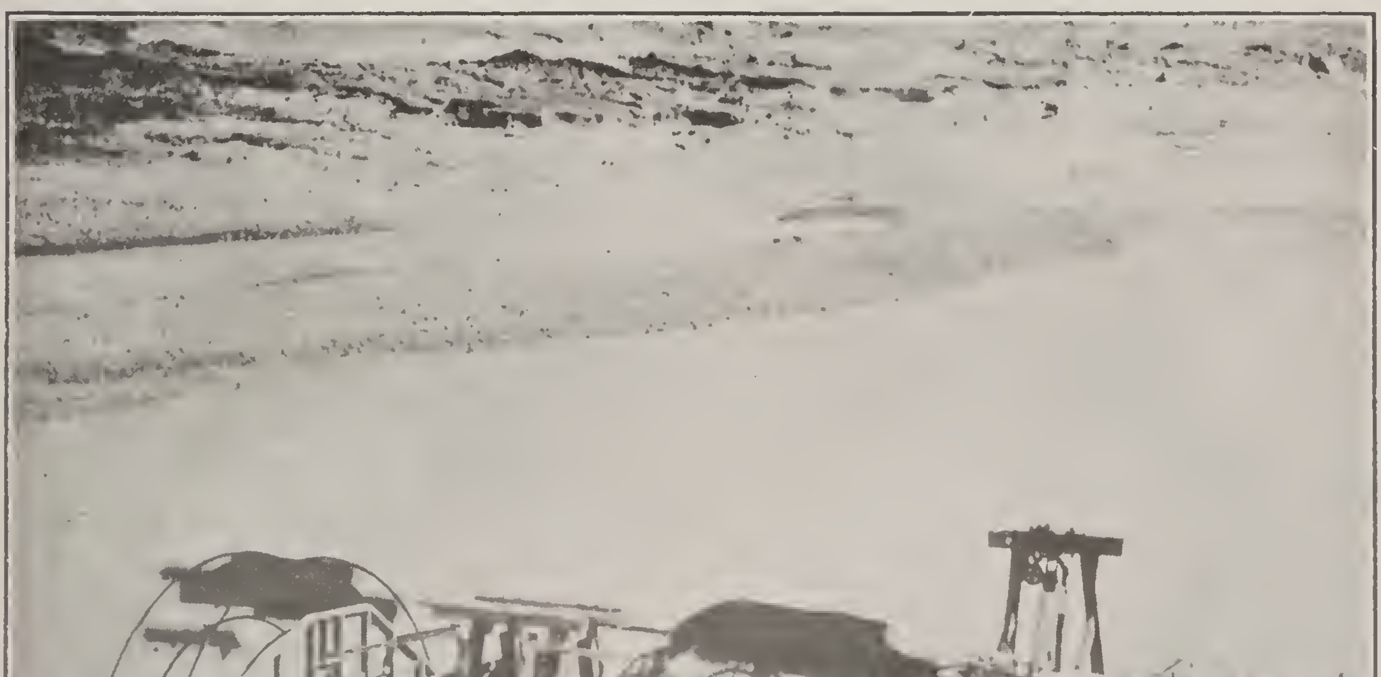
Although dredging for gold had its inception in New Zealand and the number of dredges at work in that country has been in excess previous to 1914 of the total number annually working in the United States, excluding Alaska, progress in dredge construction and operation was far more rapid in the United States than in New Zealand. The improvement in this country no doubt has been due largely to the progressive spirit of operators and the keen competition that early developed among dredge-construction firms. The latter strove to build machines of increasing efficiency for a rapidly growing demand, and the operators desired machines of greater capacity and strength.

DREDGING IN CALIFORNIA AND MONTANA.

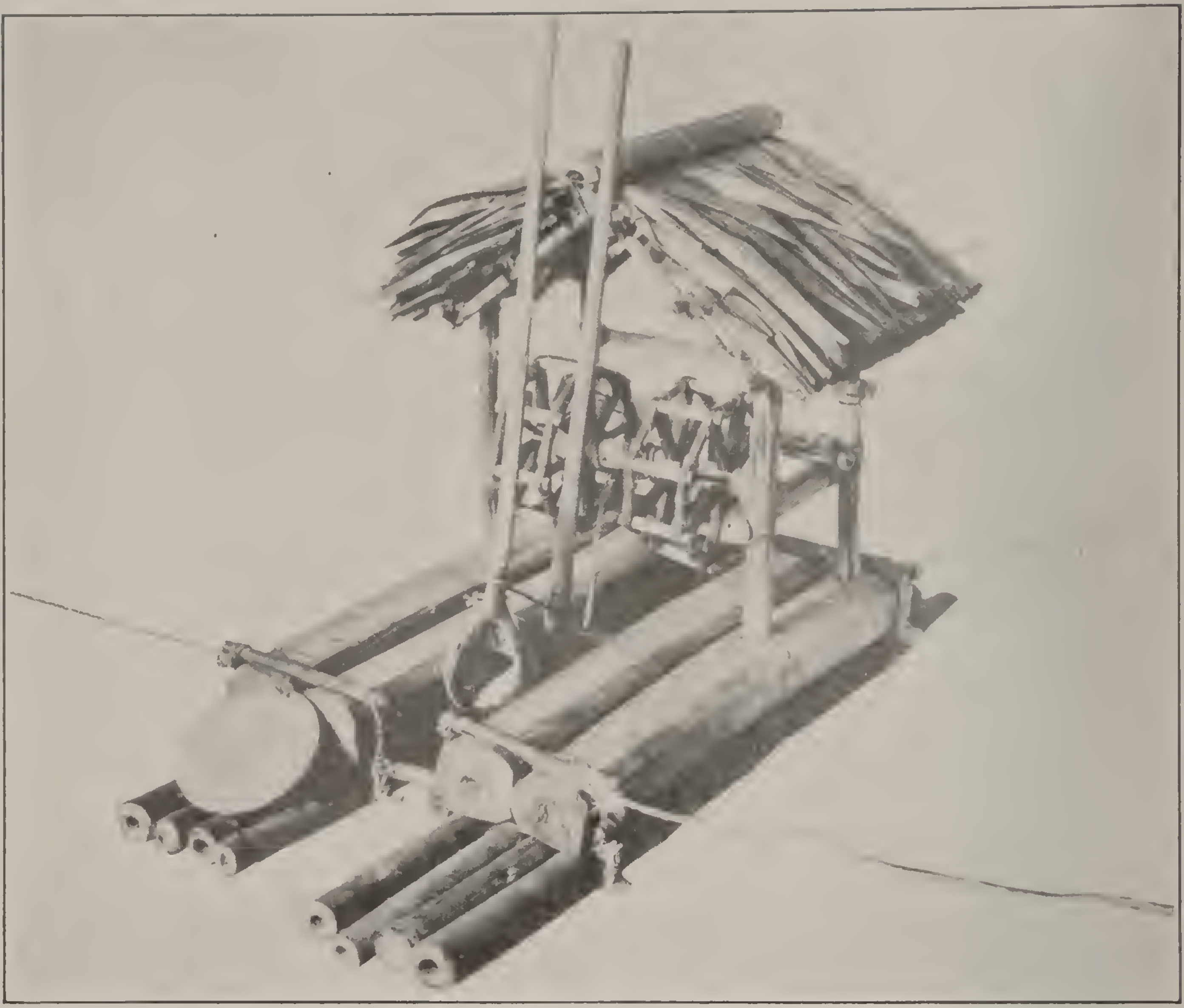
The first successful bucket-lift gold dredge in the United States began work at Bannack, Mont., in 1894, and in the total production of gold won by dredging in this country, Montana is second only to



A. SPOON DREDGE, NEW ZEALAND, 1865.



B. CURRENT-WHEEL DREDGE, NEW ZEALAND, 1882.



A. MODEL OF DREDGE USED BY NATIVES IN PHILIPPINE ISLANDS.

B. FACE OF CUT IN GROUND ALONG MOKELUMNE RIVER, CAL.

California. The output from gold dredges in California has far exceeded the combined output in all other States and, for that matter, in any foreign country. Also, it is true that most of the progress made in dredge construction has been due to dredging operations in California, although Montana has contributed materially to the development of the modern gold dredge. A number of the features that originated or were first tried out in Montana are discussed in a paper by Jennings.[a]

The history of successful gold dredging in California practically begins with the floating of the first bucket dredge that proved a success in that State at Oroville, March 1, 1898. A dredge designed after the type used in New Zealand was built by R. H. Postlethwaite on the Yuba River in 1897, and had the stream been less turbulent that dredge also would have proved a success, as similar boats operated later in other fields. However, because of difficulties not at first realized the dredge sank to the bottom soon after starting. A number of other dredges were subsequently built by the Risdon Iron Works after the design of Postlethwaite and were more or less successfully operated for many years in California. These early Risdon dredges were all modeled after the New Zealand dredges of that period.

Figures showing the number of dredges operating and the value of the gold recovered by dredging in California from 1896 to 1915 follow:

Value of gold won by dredging in California, 1896–1915.[b]

Year.	Number of dredges.	Value of gold.	Increase (+) or decrease (−).
1896	1	[c] $2,000	
1897	1	[c] 2,000	
1898	3	18,887	
1899	8	206,302	+ $187,415
1900	16	200,929	− 5,373
1901	22	471,762	+ 270,833
1902	29	867,665	+ 395,903
1903	31	1,475,749	+ 608,084
1904	42	2,187,038	+ 711,289
1905	50	3,276,141	+1,089,103
1906	59	5,098,359	+1,822,218
1907	57	5,065,437	− 32,922
1908	69	6,536,189	+1,470,752
1909	63	7,382,950	+ 846,761
1910	72	7,550,254	+ 167,304
1911	65	7,666,461	+ 116,207
1912	65	7,429,955	− 236,506
1913	63	8,090,294	+ 660,339
1914	60	7,783,394	− 306,900
1915	58	7,796,465	+ 13,071
Total		79,104,231	

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[a] Jennings, Hennen, Gold dredging in Montana, with a chapter on placer mining methods and operating costs, by Charles Janin, Bull. 121, Bureau of Mines, 1916, 63 pp.

[b] Figures for 1898–1914 from Mineral Resources U. S. for 1914, U. S. Geol. Survey, pt. 1, 1915, p. 357; figures for 1915 from Mineral Resources U. S. for 1915, U. S. Geol. Survey, 1916, pt. 1, pp. 211, 213.

[c] Estimated.

A short description[a] of the first Yuba River dredge is inserted here as of interest in comparison with the modern dredges now working on that river and described in other chapters of this bulletin:

> The dredge consists of two long pontoons, each 96 feet long by 9 feet beam. These are connected at the stern by a small pontoon 17 feet long and 5 feet wide, the bow being connected by heavy overhung beam. This practically makes one boat 96 feet long, 23 feet wide, with a well-hole 5 feet wide, running through the center for some 75 feet.
>
> The ladder, 67 feet long, built up as a heavy lattice girder, is hung at the stern end by a bar fixed across a heavy wooden framing. The lower end of the ladder carries a five-sided tumbler and is suspended by blocks and tackle to a cross beam. By means of wire rope and blocks the winch can raise or lower the bottom end as required. The top tumbler is carried by the timber framing some 3 feet above the top end of the ladder. The continuous bucket chain comes up the top side of the ladder on rollers around the top tumbler and back in a catenary curve to the lower tumbler. The top tumbler is driven through a rope transmission and heavy gears by the engine, a vertical compound condensing one, which also drives the pump and indicates about 35 horsepower.

This dredge was designed to dig 45 feet deep, whereas the modern dredges on the Yuba dig 70 to 85 feet below water level, a feat that was considered by come engineers mechanically impossible, even as late as 1905.

Up to 1898 gold dredging in the United States had not been financially successful, though dredges embodying the ideas of various manufacturers were tried in California, British Columbia, Idaho, Montana, and Colorado. From crude beginnings, however, the machines were rapidly improved until in some localities, as at Bannack, Mont., the efforts of the operators began to meet with greater financial encouragement. Most of these early dredges were light, relatively inexpensive affairs, costing $20,000 to $30,000 and were equipped with buckets having a capacity of 2½ to 5 cubic feet. The first Yuba River dredge is stated to have cost $25,000; a boat now being built in that district will cost about $450,000.

BUCKET-ELEVATOR DREDGE.

The first successful bucket-elevator gold dredge to operate in the United States was built in Montana. It was known as the *Fielding L. Graves*, and was constructed on Grasshopper Creek near Bannack. G. T. Reiling, now interested in gold dredging at Breckenridge, Colo., was connected with this early enterprise in the spring of 1895. The dredge was of the double-lift type. The material, excavated and elevated by the buckets as in the present single-lift type, was discharged into a revolving screen with large perforations, so that all gravel over 4 inches in size was discharged overboard at the side of

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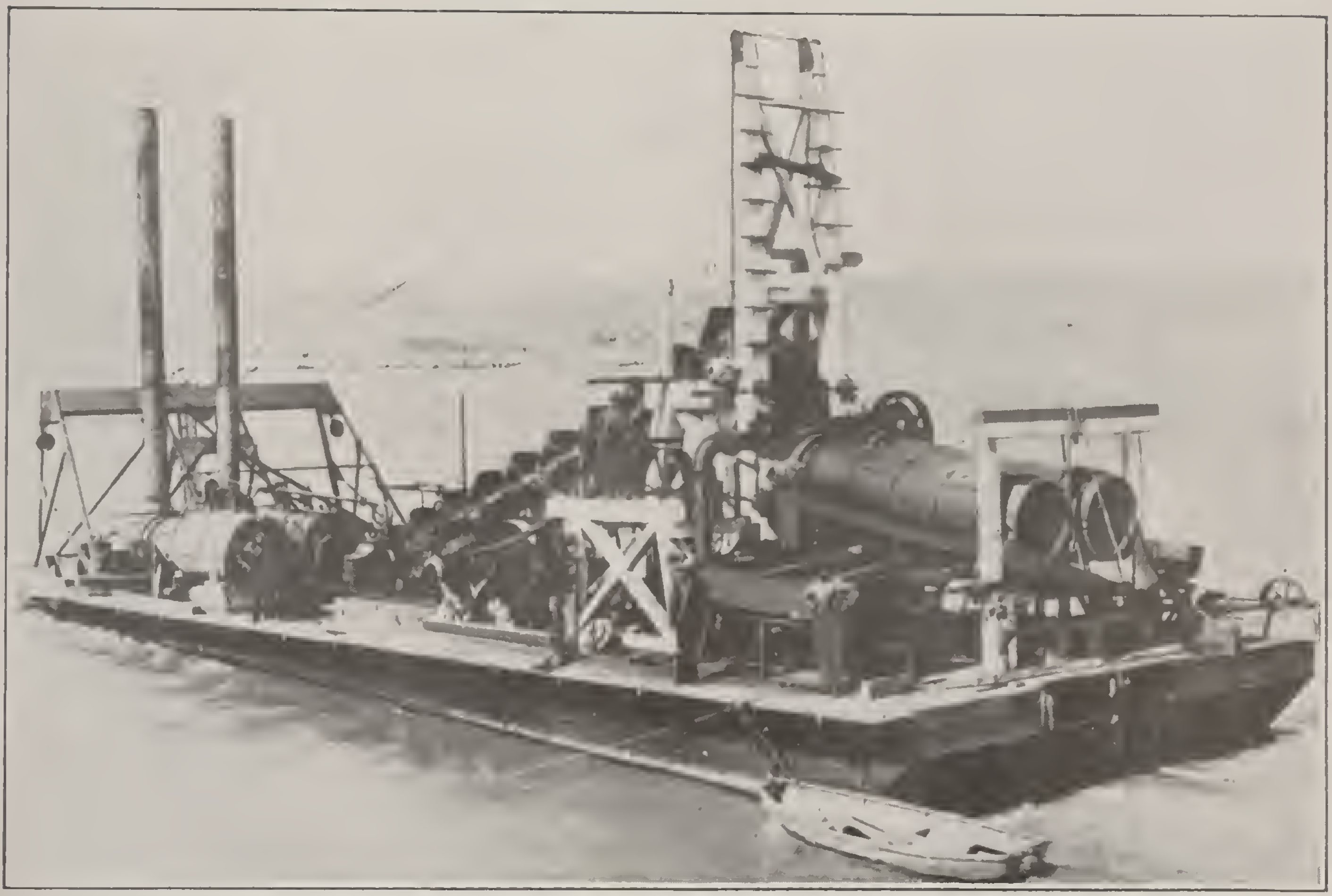
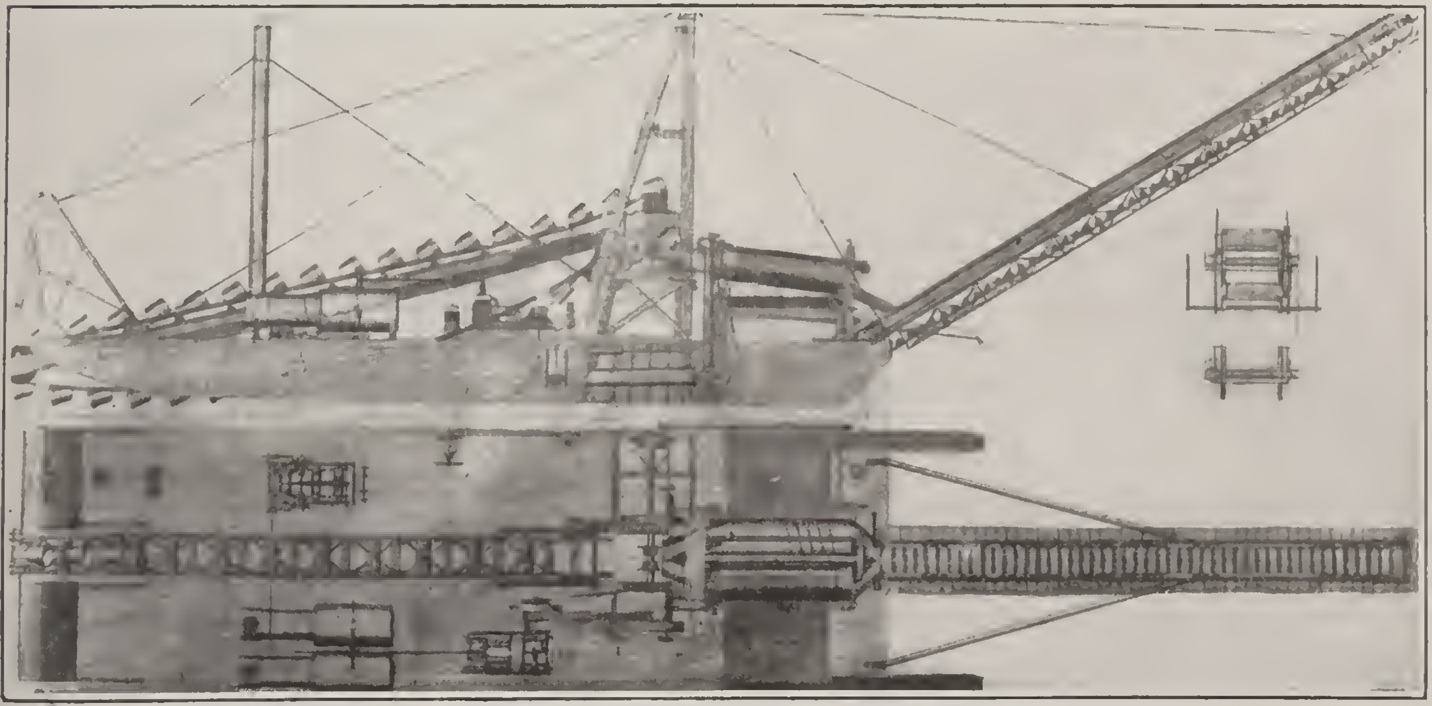
[a] Postlethwaite, R. H., River dredging for gold, Min. and Sci. Press, vol. 75, Sept. 4, 1897, p. 217.



A. NEW ZEALAND DREDGE USING TWO SCREENS.



B. EARNSCLEUGH NO. 2 DREDGE, CLUTHA RIVER, NEW ZEALAND.

the dredge near the center while the fines passed to a sump. A second line of buckets lifted the material from this sump to a sluice fitted with riffles. The gold was caught on these riffles, which at times were charged with quicksilver. The tailings were discharged about 100 feet astern of the boat.

Several points of interest about this dredge are not generally known. The second lift was by elevator buckets, electricity was the motive power, and the boat was equipped with two spuds. The washing sluice or flume was at first suspended from cables anchored to towers on either side of the gulch; but as this method proved impracticable, it was abandoned and the sluice placed on an auxiliary scow. Plate IV, *A*, shows this dredge as it now appears.

The *A. E. Graeter*, the second dredge to work in the Bannack district, was built in 1897. This dredge, also illustrated here (Pl. IV, B and *C*), although it had buckets of the same size as those on the *Graves*, differed from that boat in having steam power and using a centrifugal pump to elevate the fines from the sump to the sluices. A point of interest in the *Graeter* is that the lower end of the tailing sluice was arranged so that it could be swung laterally by means of cables, in order to deposit the fines over a wider area. To preserve the balance of the boat, which might be affected by this change of dumping point, two tanks each 42 feet long, 12 feet wide, and 42 inches deep, were contained in either side of the hull. When the sluice was to be swayed to one side water was pumped into the tank on the opposite side.

Plate V, *A*, shows one of the early Colorado dredges that worked in the Breckenridge district in 1898, the forerunners of the dredges now successfully operating there. It is interesting to study the different forms of construction used in these early boats and to note the gradual development of mechanical and structural improvements.

PROGRESS IN DESIGN AND EQUIPMENT.

The work of the dredges built prior to 1900, or even of some built as late as 1910, was by no means completely satisfactory from the start, and large sums of money were expended in changes and experiments before some of the ventures proved a financial success. Troubles with the later dredges were caused partly by the operators becoming more ambitious, as early difficulties were overcome, and attempting to work deposits that at first were considered entirely outside the realm of possible dredging. One can picture the astonishment of the pioneers in this industry when expressing anxiety over the work of their early boats, capable of handling only 50 cubic yards an hour, if they had been asked to design and finance a dredge that would dig 80 feet below water level and would handle

500 to 600 cubic yards an hour or 400,000 cubic yards a month for months at a time. Yet this is what the mammoth gold dredges in California, Idaho, and Montana are doing now. Too much credit can not be given the pioneers for their persistent and progressive efforts, and it seems eminently fitting that W. P. Hammon, who has been the leading operator since gold dredging began in California, is still actively interested in the operations of the largest gold-dredging company in America. Among other names prominent in California dredging are those of R. G. Hanford, George Carr, Frank Griffin, Charles Gardiner, and Louis Hopfield; in Montana the names of N. S. Schafer, Gordon McKay, and Charles Kammerer are well known. In addition to these men many others have contributed greatly to the development and success of the modern gold dredge.

GOLD PRODUCED BY DREDGING.

The total production from gold by dredging in the United States, excluding Alaska, from the beginning of operations up to the end of 1914, has been approximately $85,000,000. Of this sum California has contributed more than $71,000,000. Montana comes next, with $6,500,000, and is followed by Idaho and Colorado, with about $3,000,000 each. During 1914 there were nearly 80 dredges busy in the United States outside of Alaska. The following table, showing the production of gold by the various States for different years, has been compiled by the United States Geological Survey:[a]

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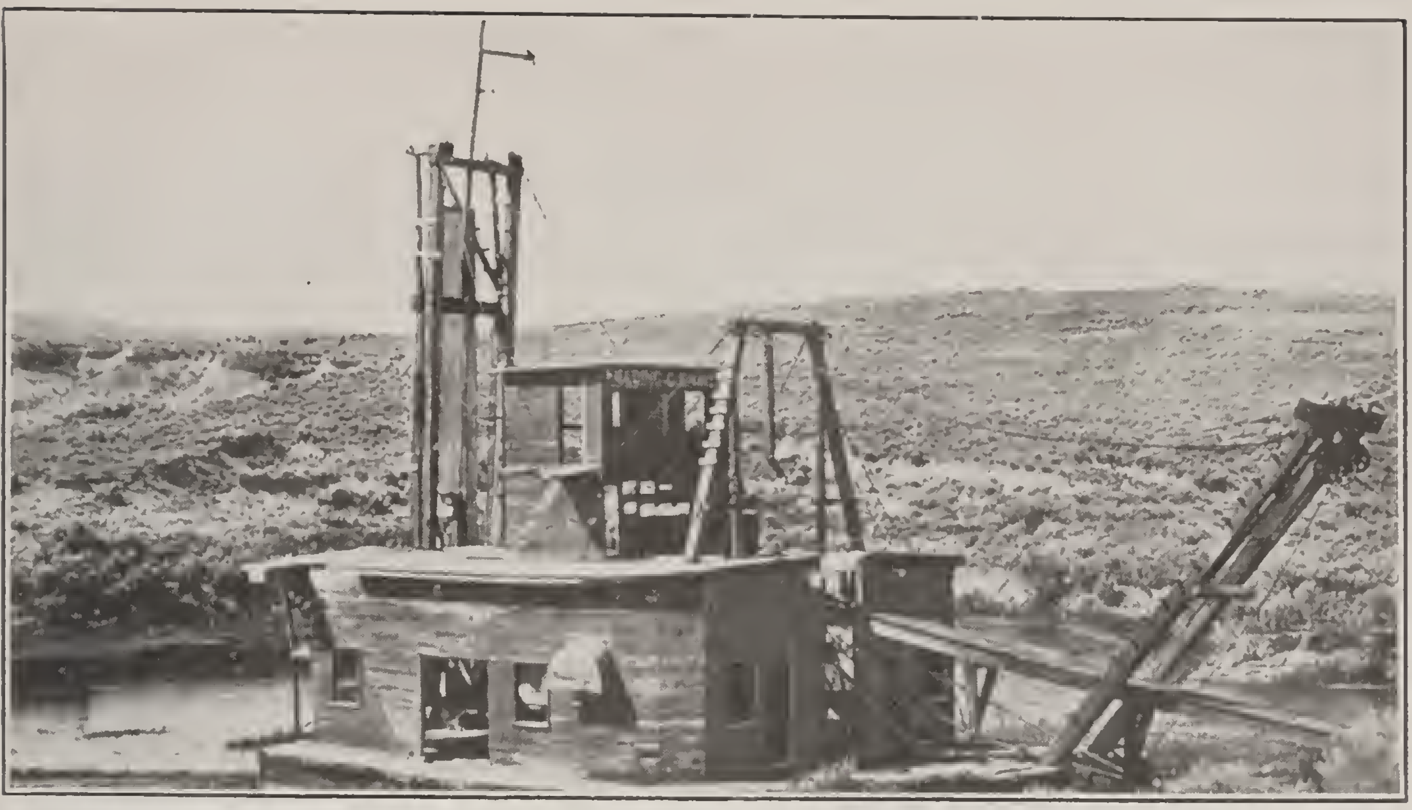
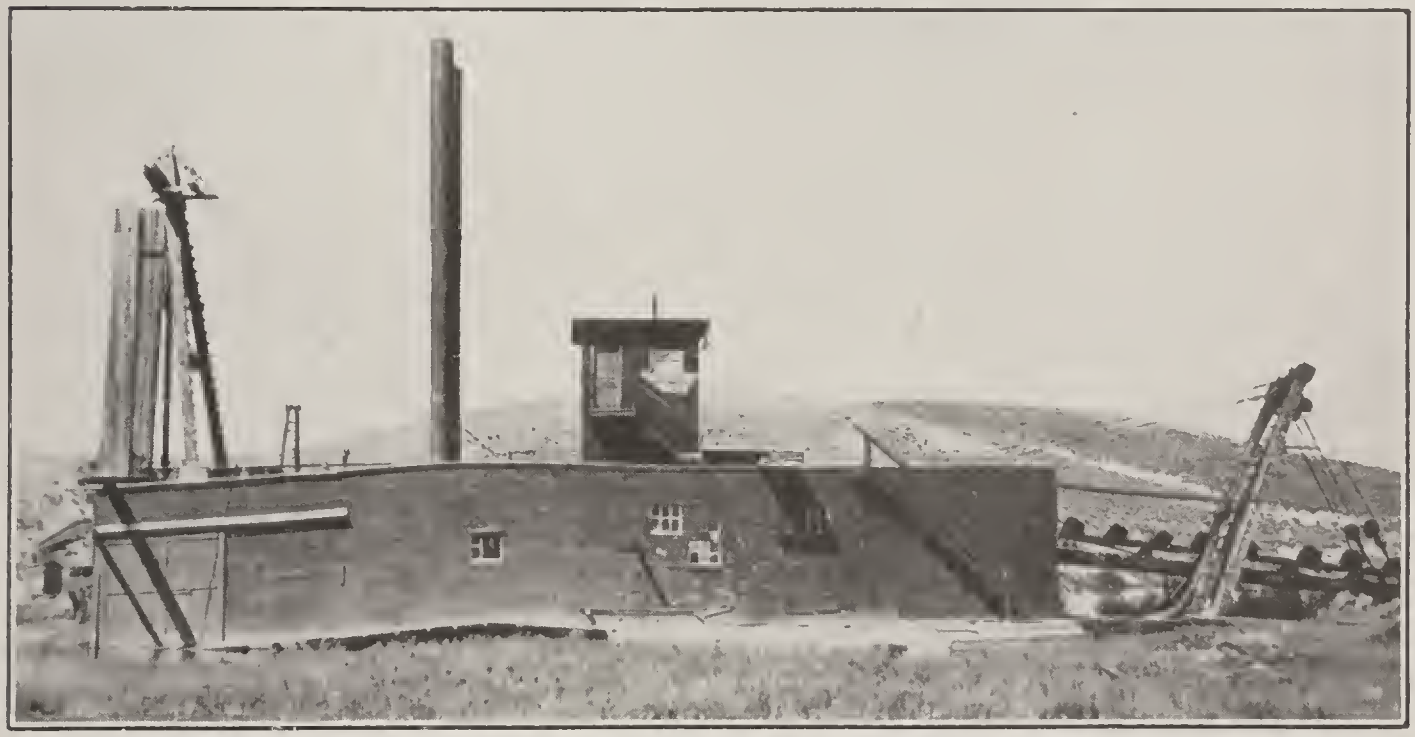
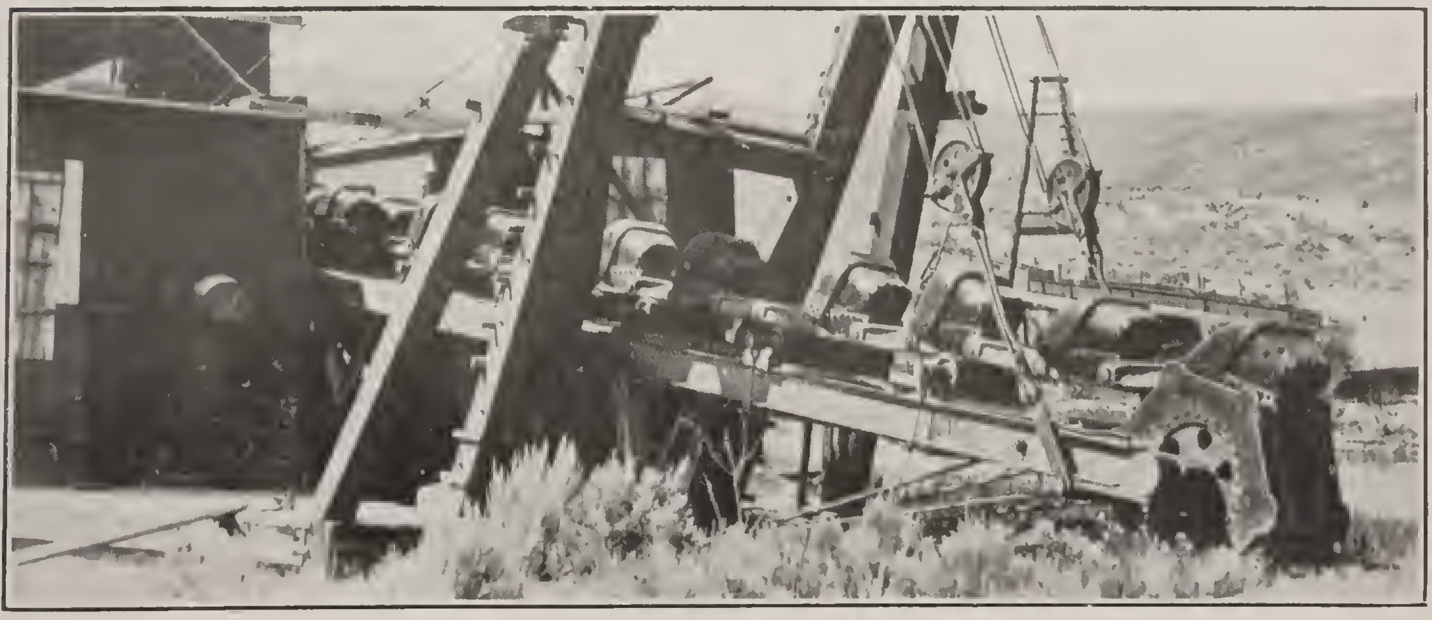
[a] McCaskey, H. D., Gold and silver (general report): Mineral Resources U. S. for 1914, U. S. Geol. Survey, 1915, p. 855.



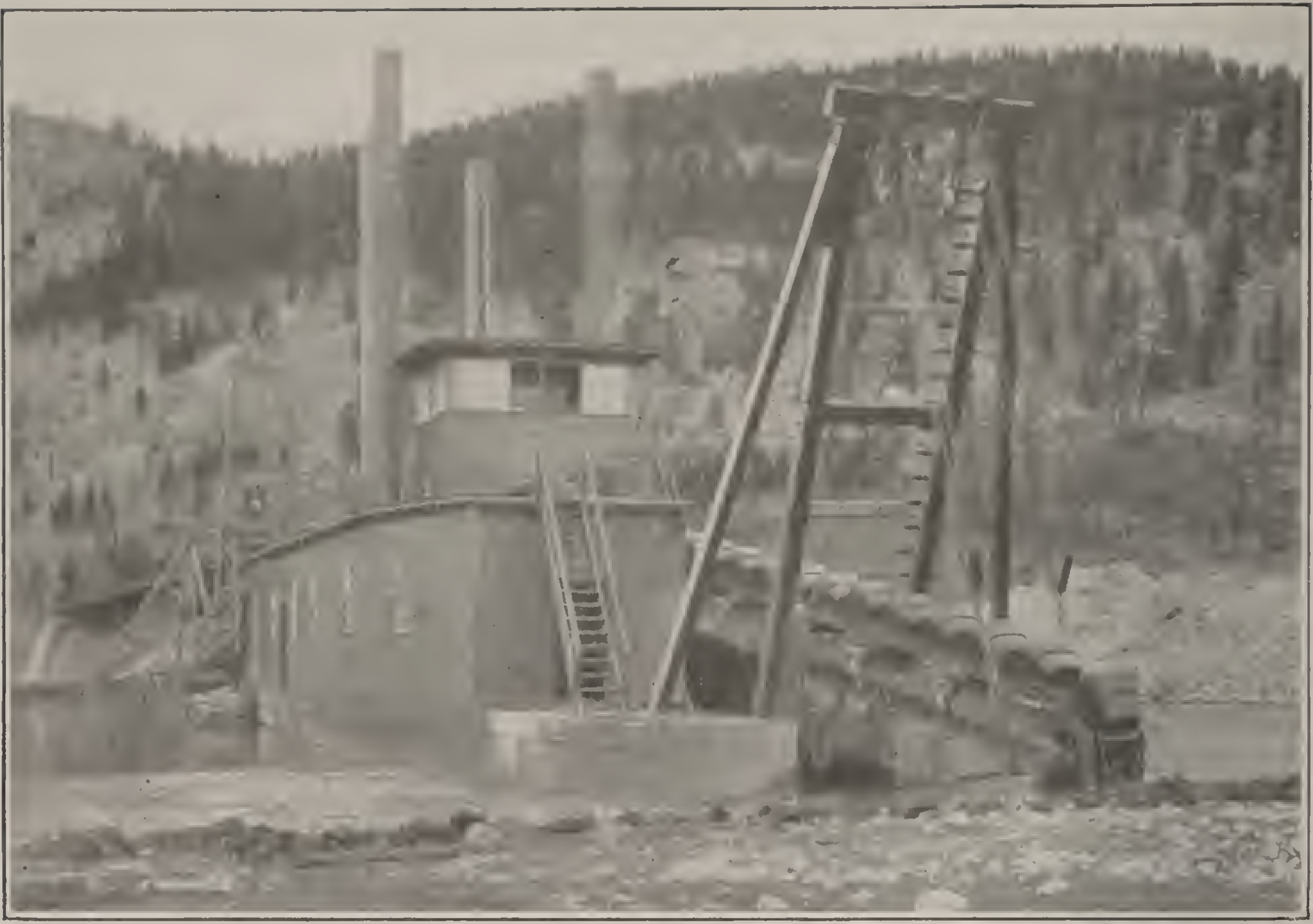
A. THE FIELDING L. GRAVES, FIRST SUCCESSFUL BUCKET-LIFT DREDGE IN THE UNITED STATES, 1894.



B. SIDE VIEW OF A. F. GRAETER DREDGE USED AT BANNACK, MONT.; STARTED 1895.



C. BUCKET LINE ON A. F. GRAETER DREDGE.



A. DREDGE BUILT AT BRECKENRIDGE, COLO., IN 1898.

B. OLD METHOD OF WORKING PLACER GROUND BELOW OROVILLE, CAL. FROM AN OIL PAINTING.

Production of gold in the United States by dredges and number of dredges producing, by States, 1896–1914.

State or Territory.	1914		1913		1912		1911		1910		1909		1908		1907		1906		1905	
	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.
Alaska	$2,350,000	42	$2,200,000	36	$2,200,000	38	$1,500,000	27	$800,000	18	$424,993	14	$170,901	4	$250,000	4	$120,000	3	$40,000	3
California	7,783,394	60	8,090,294	63	7,429,955	65	7,666,461	65	7,550,254	72	7,382,950	63	6,536,189	69	5,065,437	57	5,098,359	59	3,276,141	50
Colorado	602,655	5	372,288	4	384,748	3	272,173	4	344,211	6	404,636	4	141,773	4	35,235	3	48,343	3	33,342	3
Idaho	568,989	4	561,876	6	481,077	8	258,791	7	91,247	6	101,704	8	77,189	5	74,438	6	38,340	3	34,336	3
Montana	835,615	5	685,210	5	710,387	6	597,778	8	473,318	7	426,649	5	402,667	4	197,141	4	397,030	4	275,542	5
Oregon							14,575	3	34,010	6	42,667	2			23,191	2				
Other States	[a] 372,130	4	[b] 317,268	2	[c] 12,744	4	[d] 16,591	5					[e] 24,852	2	[f] 10,260	3	[g] 19,322	4	[h] 28,015	4
Total	12,512,783	120	12,226,936	116	11,218,911	124	10,326,369	119	9,293,040	115	8,783,599	96	7,353,571	88	5,655,702	79	5,721,394	76	3,687,376	68

State or Territory.	1904		1903		1902		1901		1900		1899		1898		1897		1896		Total value.
	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	Value.	Dredges.	
Alaska	$25,000	3	$20,000	2															$10,100,894
California	2,187,038	42	1,475,749	31	$867,665	29	$471,762	22	$200,929	16	$206,302	8	$18,887	3	$5,000	1	$2,000	1	71,314,766
Colorado	65,594	3	15,000	1	10,000	1	6,000	1											2,735,998
Idaho	99,110	7	86,113	6	101,257	10	116,117	6	129,443	6	62,436	5	13,920	1	11,436	1			2,907,819
Montana	245,700	2	229,332	4	318,914	7	146,134	5	185,665	5	165,440	5	154,893	4	102,120	4	42,000	1	6,595,535
Oregon																			[k] 114,443
Other States	[i] 101,275	4	[j] 89,870	1	[j] 71,686	1													[l] 1,064,013
Total	2,723,717	61	1,916,064	45	1,369,522	48	740,013	34	520,037	27	434,178	18	187,700	8	118,556	6	44,000	2	94,833,468

[a] Oregon, Nevada, Alabama, North Carolina.
[b] Oregon, Nevada.
[c] South Dakota, North Carolina, Georgia, Wyoming.
[d] Nevada (2), South Dakota, Georgia, North Carolina.
[e] Georgia, Oregon.
[f] Georgia (2), New Mexico.
[g] Georgia, Oregon.
[h] Georgia, North Carolina, Oregon, New Mexico.
[i] Oregon (3) New Mexico.
[j] New Mexico.
[k] Part of output only; see other States.
[l] Largely from Oregon.

The significance of these figures is emphasized by comparing them with returns from foreign countries as shown by the report on dredge mining in Victoria for 1914.[a] In Victoria the total production of gold by dredges from the beginning of operations to the end of 1914 was 1,094,562 ounces or approximately $22,000,000. The largest number of dredges busy in a single year was 133 in 1897, when 20,000,000 cubic yards of material was handled with a reported average recovery of 2.4 grains or approximately 9.6 cents a yard. Compare these figures of yardage with that of one company, the Natomas Consolidated, of California, which in 1914 operated 13 dredges that handled in all 25,800,000 cubic yards with an average gold recovery of 10.25 cents a yard. In New South Wales according to the reports of the Department of Mines, during the years 1900 to 1914, inclusive, the total yield for gold dredging bucket and pump dredges was £1,682,540, or about $8,400,000. During 1914 there were 18 bucket dredges and 8 pumping plants employed, and the 18 bucket dredges treated 4,315,962 cubic yards of material during the year. These figures of dredge capacity might be compared with those of the Boston & Idaho Co. at Idaho City, Idaho, where an 18-foot bucket dredge handled a total of 4,670,124 cubic yards in 1914.

According to the minister of mines of New Zealand,[b] 64 gold dredges in 1914 produced gold to the value of £191,112. During the past decade the number of gold dredges decreased from 167 in 1906 to 64 in 1914, and the value of the output from £500,000 to that given for 1914. From 1906 to 1914, inclusive, the production of gold by dredging in New Zealand was £2,883,570, the average annual recovery being less than £5,000 per dredge.

Such figures show how the first efforts in other countries have become the basis of an industry that has been developed in the United States beyond all dreams of the pioneers.

As the modern gold dredge is the result of gradual development, the attaining of the present mechanical efficiency and economic success has required many years. After the spoon dredge operated by hand came the steam shovel, then came the endless bucket-chain, first with buckets and connecting links and then, years later, with close-connected buckets. At first the power was steam, then current wheels, then steam again. Some dredges obtained power from water wheels on the boat, the water under pressure being brought through pipes with flexible or swivel joints. The application of electric power generated by central steam plants or by hydroelectric plants was a great advance in dredge operation. Gas and oil engines of various types and makes for dredge purposes were tried and successfully developed from experience in Alaska. In some foreign fields producer gas has been used to a small extent but has not proved a great success. One of the most recent innovations in power installation is on a dredge for the Philippines, which carries a steam turbogenerator for electric power. From the spoon dredge in 1865 to the first elevator-bucket dredge in 1882 was a long step, but the advance from the early bucket dredges costing $20,000 to the present mammoth dredges costing $400,000 is fully as great.

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[a] Victoria, Australia, Department of Mines, annual report for 1914, p. 7.

[b] New Zealand, Mines statement for 1914, 1915, p. 23.

DREDGING IN COLD REGIONS.

The pioneers of the industry in the far north, in addition to the ordinary problems encountered in gold dredging, have been obliged to overcome the seemingly insurmountable difficulty of dredging frozen ground. The successful use of steam for thawing such ground has been perhaps the most important development of dredging in the north and has greatly extended the dredgeable areas in Alaska and Yukon Territory.

The first dredge to work in Yukon Territory was built on Lewis River in 1899. In Alaska the first dredge on the Seward Peninsula was built on Snake River near Nome in the same year. Since then a number of dredges with 16-foot buckets have been built on the Yukon and dredges with 10-foot buckets on Seward Peninsula, the total number of dredges operating in both fields being about 54 in 1914.

The production of gold by dredges in Alaska has exceeded $10,000,000. In Yukon Territory to the end of 1914 the production approximated $20,000,000, one American company, the Yukon Gold, contributing more than $16,000,000.

EVOLUTION OF GOLD DREDGES.

The modern dredge can not be claimed as the child of any one brain. Rather it is a composite product, representing the progressive thought and earnest cooperation of many individual operators, engineers, and manufacturers. Even some of the many utterly unsuitable and freak machines tried in the early days may be said to have suggested an improvement made in the practical dredges of that time, nor can one say that no further improvements are coming and that perfection has been reached. Some comment on the direction in which improvement is possible will be found in the chapter on dredge construction.

GOLD-DREDGING DISTRICTS IN THE UNITED STATES.

The principal gold-dredging districts of the United States are in California. Montana comes second in importance and production; Colorado, with two proven districts, is third; and Idaho and Oregon practically complete the list of the States. Some dredging has been done in Nevada, New Mexico, and Georgia, but the operations have been relatively of small economic importance.

In Alaska the principal dredging operations have been in the Nome and Council districts on the Seward Peninsula. Others are scattered in the Forty-mile, Iditarod, Ruby, Circle, and Fairbanks districts In the Philippines, the best known and most important district is the Paracale. The new dredge at Mambulao may open another field of importance.

CALIFORNIA.

OROVILLE AND HAMMONTON DISTRICTS.

In California the oldest and best known district is Oroville, but it is being rapidly worked out and the number of dredges is decreasing from year to year. In 1910 there were 25 dredges in the Oroville district, as compared with 8 at present.

As regards production, the most important district is at Hammonton on Yuba River. In 1915 15 dredges were busy there. The value of the gold recovered in 1910 was $3,172,476; in 1915 the value was $2,675,090. The decline in production is due to the working of poorer ground, for the total yardage handled by the dredges has increased. This fact may be made plainer by comparing the figures for one company, the Yuba Consolidated. In 1911 the 12 dredges of this company handled approximately 14,000,000 cubic yards, whereas for the year ended February 28, 1916, the same number of dredges handled about 17,750,000 cubic yards; yet the gross production for 1915 was about $800,000 less than that for 1911. New dredges for Yuba County, which will start in 1917, include four 16-foot bucket, all-steel boats for the two companies now operating and a 9-foot boat for a new part of the district. Yuba County will lead in 1917 in gold production and number of dredges operating.

OTHER DISTRICTS.

In Calaveras County, at Comanche, and Jenny Lind there were 5 dredges busy in 1915. In Placer County there were 3 dredges operating in 1915. The principal district is on the American River, where the extent of the available ground is limited. In 1910 dredging was confined to the Bear River, where it proved unprofitable.

In Sacramento County the Folsom district is still a heavy producer. In 1915 there were 11 dredges busy there as compared with 8 at the beginning of 1910. This district produced from dredging in 1915 $2,129,781 in gold. The reconstruction of several old boats to embody some important changes was undertaken in 1916.

In Stanislaus County one dredge has been in operation since 1908 and in Merced County one since 1907. In Siskiyou County there are two dredges busy, which in 1915 produced about $211,000 in gold. In Trinity County there are two dredges, one at Trinity Center and another at Lewiston. In 1916 considerable prospecting was done near Carrville and Junction City, and several new dredges are under construction for work in this county.

In Shasta County successful gold dredging is comparatively recent, although a number of attempts in different areas were made some years ago. Dredges are working on Cottonwood Creek and on the Salzer ranch, near Redding. The Igo district has been recently prospected.

PRESENT STATUS OF DREDGING.

As a whole, the output of gold from dredges in California has reached its maximum. The Hammonton and Folsom districts seem to have passed their highest mark, the Oroville district is nearly exhausted, and the possible area of more recent development in the northern counties is very limited. To offset the exhaustion of virgin territory, several attempts have been made to handle the tailings from early dredges, and it is possible that some plan will be evolved whereby the reworking of dredged ground will become profitable. The recent construction of a number of large dredges has greatly increased the yardage handled in California, though the gold content of the gravel is far less than of that worked in past years. With the operation of dredges now being built and the addition of others, the gold won by dredging in California will be of considerable importance for years to come.

MONTANA.

The four dredges of the Conrey Mining Co. at Alder Gulch, in Madison County, comprise all those in the State. These dredges have been working successfully a number of years, and, with the exception of the California districts named above, the district is the largest producer in the United States.

This district is being rapidly worked out, and one dredge has been shut down since the preceding statement was written.

COLORADO.

In the Breckenridge district, in Summit County, four dredges are at work. The Derry Ranch dredge, near Leadville began work in 1915. In addition to these, the possible operation of dredges in other districts is indicated by considerable prospecting done at different times.

IDAHO.

Dredging still continues in the Idaho City and Kirtley Creek districts, as well as minor operations in less important districts; and the production has increased steadily until it is now about six times what it was in 1910. At present four dredges are at work in this State. These fields are rapidly nearing exhaustion, but some new areas are now being equipped with dredges. One of the dredges of the Yukon Gold Co., having worked out the ground for which it was built, has been moved from the Yukon territory to Idaho.

OREGON.

As dredging on the Powder River had proved successful, another dredge was installed during 1916, making two for this district. The Powder River is the only district where active dredging is in progress in the State, though a dredge has recently started work in the John Day Valley.

ALASKA.

In 1915 there were 42 dredges reported operating on the Seward Peninsula. Most of these have small capacity and shallow digging depth. Two of the dredges are recovering tin ore. In 1915, a dredge was built in the Ruby district, on the lower Yukon, and one near Circle City. The small dredge on Fairbanks Creek, brought from the upper Yukon to the Tenana, is still in operation.

Two dredges are at work in the Iditarod district; and others are in prospect. The total production of gold by dredges in 1915 was about $2,330,000.

PHILIPPINE ISLANDS.

In 1915 six dredges were busy in the Philippines. Four more had been ordered or were being constructed. Most of the active dredges, all but one in fact, are working in the Paracale district, Luzon. The other one, at Mambulao, is a new dredge recently constructed, but no reports on its operations are as yet available. The annual production of gold from dredging in the islands is reported to be upwards of $1,200,000.

TOPOGRAPHY AND GEOLOGY OF DREDGING AREAS.

The largest dredging fields in California, and, so far as production is concerned, in the world lie along three of the largest rivers draining the principal mineral region of the State. The mineral area drained by the different branches of these rivers is about 350 miles long and 30 to 70 miles wide, and includes many of the ancient river channels that were mined for gold with considerable profit in early days. The present river systems have somewhat different courses and frequently cut across these ancient channels (Pl. II, *B*). River bars and flood plains constitute for the most part the placer deposits of the present streams. The placer gold is from the disintegration of the old auriferous slates and other rocks and the lodes they contained and also from ancient channels eroded by the present river.[a]

CALIFORNIA DREDGING DISTRICTS.

The predominating conditions in California are probably typical to a large extent of those in other fields.

Undoubtedly the placers derive their gold from eroded vein and other deposits in the bedrock. One should remember that a good placer should contain fairly coarse gold, because fine gold does not concentrate as readily and is borne away with the sands. Quartz veins that carry mainly coarse gold are apt to be "spotty" and uncertain. In the really valuable quartz lodes the gold mainly is fine, either free or in pyrites, and the erosion of such lodes does not result in rich placers. For this reason there is not likely to be any close association between the best quartz mining districts and the most productive placers. The American and Feather Rivers in Placer and Plumas Counties, Cal., have furnished rich placers but are notably poor in quartz mines. The Caribou district of British Columbia and the Klondike have similar characteristics. The Mother Lode region of California has yielded very little placer gold. The Columbia placer district of Tuolumne County, though limited, was one of the richest in California. It adjoins a district that has pockety quartz mines that produced considerable coarse gold—notably the Bonanza Mine, which was profitable at times—but has shown no lodes that would maintain profitable mining on an extensive scale.

All the gold of the valley placers is fine as compared with the coarse gold found along the higher courses of the present rivers. Although the finest particles have undoubtedly been carried some distance, the coarse gold has probably not traveled far from where it first lodged.

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[a] Lindgren, Waldemar, Tertiary gravels of the Sierra Nevada of California, U. S. Geol. Survey Prof. Paper 73, 1911, 226 pp., 28 pls.

Most of the gold is in fine flat scales that are comparatively easy to recover; but some is as an almost impalpable powder and is carried off in suspension and some is associated with sulphides, or is coated with iron or other oxides, and is lost through failure to amalgamate. Occasionally fair-sized nuggets are found in the dredge sluices and at Breckenridge, Colo., 2½-inch perforations in the lower sections of the revolving screen were necessary to prevent a loss of gold nuggets. At the Alta Bert property in Trinity County, Cal., a large part of the gold is relatively coarse.

CHIEF DISTRICTS.

The principal dredging districts in California are at Oroville on Feather River; at Daguerre Point, about 12 miles above Marysville, on the Yuba; and at Folsom, on the American River. A smaller district has been worked for some time near Chico in Butte County, and some years ago dredging was tried about Wheatland on the Bear River, but the gravels proved to be poor and operations were discontinued.

Smaller quantities of gold are dredged in Siskiyou, Trinity and Shasta Counties. Dredges are also busy in Calaveras County, Merced County and in Stanislaus County, and some dredging is being done on the American River above Auburn.

The depth of the gravels ranges from 15 to 80 feet. In part the gravels are coarse, cobbles up to a foot in diameter being by no means uncommon. In Trinity County, Cal., many of the bowlders met weigh more than a ton. The gravels contain much sand and clay in places and are ill sorted. The Oroville gravels almost universally rest on a false bedrock of volcanic tuff or hardpan; some of them lie below the present river level, and others rest on benches up to 100 feet above the channels of to-day. The tuff forms a sheet 12 to 60 feet thick. At Oroville, and along Yuba River, there is auriferous gravel below it, but under present conditions this lower gravel can not be worked.

While the greater proportion of the gold is in contact with or directly above bedrock, gold is by no means everywhere confined to the lower layers. Gold is often distributed through the gravels or contained in one or more upper beds. Where occasional seams of clay or other material from an impervious layer in otherwise porous gravel, the downward movement of heavy minerals is arrested, and above these layers "pay streaks" are formed, a cross section of a deposit sometimes showing several of them. Where the material is clayey, the gold is likely to be distributed throughout it. As the distance from the mouth of a canyon increases the gold becomes finer and the gravels poorer.[a]

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[a] For a discussion of the geology of the California placers, see Lindgren, Waldemar, The Tertiary gravels of the Sierra Nevada of California: U. S. Geol. Survey Prof. Paper 73, 1911, 220 pp.

OROVILLE FIELD.

The Oroville dredging ground (Pl. V, *B*), now practically exhausted, comprised about 7,000 acres of the plain of Feather River, beginning at the town and continuing to about 5 miles below, where the ground gradually became poorer. The width varied from less than a mile to about 2 miles. The average depth of gravel is about 30 feet, increasing to 40 farther down the river. The gravel rests on a false bedrock consisting of a compact tuff containing no large lava pebbles or fragments.

The Feather River, along which the placers lie, flows through a broad valley whose surface is covered with gold-bearing gravels and sands of varying richness and depth. The gold originally came from the gold-bearing quartz veins and old gold-bearing gravels in the mountainous region of the river's upper course, and has been redeposited with the gravels and sands in the neighborhood of Oroville.

As a whole the placer ground is chiefly heavy gravel, with a top layer of sand or sandy loam. This loam is underlain by a stratum of mixed gravel and sand, and in places by well-defined strata of sand, below which are other strata of coarse or fine gravel and sand, the size of the rock fragments varying from that of a pea to occasional bowlders 18 inches in diameter. This mixture of gravel and sand is 20 to 30 feet, and in places 40 feet thick. Beneath it lies the false bedrock, a tuff or volcanic ash, fairly smooth and regular in contour, and soft enough to dig with the dredge buckets. In places the placers contain cemented gravel, and in places considerable clay makes washing difficult. Most of the gold is in coarse gravel; here and there some is contained in sand. The gold is fine, and some of it may be called flour gold. The gold content of the gravel worked ranges from 8 to 35 cents to the cubic yard.

The fineness of grain of the gold in the Oroville field is given in the following statement, which represents averages from a number of tests made by screening through different sized screens:

Size of gold grains from the Oroville field.

Mesh of screen.	Per cent.
60 and coarser	58.1
60 to 100	15.3
100 to 120	2.9
120 to 150	2.4
150 and finer	21.1

In the other districts the gold has approximately the same size of grain. Occasionally, as in Trinity and Siskiyou Counties, small nuggets having a value of 50 cents to several dollars are found, but in California such nuggets are confined almost entirely to the gravels of those counties. The gold is generally of high grade, its fineness averaging 915 to 930. Platinum, iridosmine, etc., are associated with the gold, but their quantity is very small. The platinum does not amalgamate and is separated by panning the black sand of the weekly clean-up of each dredge. The production of platinum from all districts amounts to only a few hundred ounces a year.

YUBA FIELD.

The Yuba field (Pl. VI, *A*) 12 miles from Marysville, at Daguerre Point, is situated where the first "greenstone" hills of Yuba County emerge from the Quaternary covering on the south side of the river. The available dredging land on Yuba River has a total area of about 6,000 acres; it extends along the river for a distance of about 7 miles and has a width of 1 to 2 miles.

The field lies only a little above river level and is covered to a depth of 40 feet or less with the old tailings the river has brought from the hydraulic mines in the mountains. Before these tailings were deposited the low bars rose to a height of 15 to 20 feet above normal water level. Above Daguerre Point, on the north side of the river, the gravels rest on the greenstone bedrock, but elsewhere in the area they lie on a stratum of hardpan or compact clay, which on the north side of the river rises to form low rolling hills. Below the clay in places is volcanic tuff, similar to that of Oroville, and probably derived from the same source, namely, the volcano of the Marysville Buttes. The bedrock brought up by the dredges does not usually show the presence of volcanic material.

FOLSOM FIELD.

The Folsom dredging field (Pl. VI, *B*) at the mouth of the canyon of American River is the largest in the State, comprising about 13,000 acres. From Folsom, in Sacramento County, the field extends mainly along the south side of the river for about 7 miles, with a width of 1 to 2 miles, forming a wide belt of dredging ground of sandy gravel 20 to 35 feet deep. In places some clay is encountered, but the conditions at Sacramento and Mississippi and on the Kendall and Nuttall tracts are generally satisfactory for economic dredging. Like the gravel of the bars, this gravel rests on a false bedrock of white volcanic ash, which outcrops in the Orangevale Bluff, north of the river, and at other points on higher ground south of the river. Only at two places along Willow Creek, within the area actually exploited, does the real bedrock (granite and slate) appear under the gravel.

Farther south and southwest of Folsom is a higher belt of terrace gravel on which the early miners did much shallow hydraulic and drift mining. This belt appears to represent an older channel, which started at Mormon Island and Blue Wing Ravine and continued along the upper course of Willow Creek. Here the channel is 400 feet wide

A. GENERAL VIEW OF THE YUBA FIELD, CAL.

B. DREDGING GROUND IN THE FOLSOM FIELD, CAL.

and formerly was worked at several places through shafts 60 to 80 feet deep. At Rebel Hill this gravel spreads out considerably and is 50 to 70 feet deep, the volcanic ash bedrock lying about 50 feet above the river. In this area the formation consists of gravel held tightly together with clay. In places the gravel is cemented and is covered with a fine clayey soil, and for years profitable dredging was thought to be impossible there. The conditions for work are unfavorable, as the digging is hard, and the clay makes washing difficult. There are two principal pay streaks; one is 34 feet below the surface, the second, at 50 feet or more, rests on the false bedrock. At Sulky Flat and Blue Ravine the gravel is similar to that on Rebel Hill and equally hard to dredge.

In California the mining season is continuous and climatic conditions are generally favorable.

COLORADO DREDGING FIELDS.

The principal dredging fields in Colorado are near Breckenridge, Summit County, where the gravels occupy the bottoms of existing valleys. The greatest known thickness, about 90 feet, is near the Gold Pan pit at Breckenridge.[a] Along French Creek the depth to bedrock in the main channel, from Nigger Gulch down, is 45 to 50 feet. Along the Blue, between Braddocks and the mouth of the Swan, the depth in the old channel is 55 to 60 feet, and along the Swan, from Galena Gulch down, the maximum thickness is 40 to 50 feet.

The width of the gravel-filled valley bottoms is 600 to 3,000 feet along the Blue, 500 to 1,200 feet along the Swan, and 700 to 1,500 feet in French Gulch.

In general, the gravels are loose and the readiness with which they crumble when undercut by the buckets facilitates dredging. Those on the lower Swan cave more easily than those on French Creek. The gravels are generally coarse and contain hard, well-rounded bowlders. Along the Blue these bowlders range in diameter from a maximum of about 6 feet near Breckenridge to 4 feet near Valdoro. In French Gulch the large bowlders rarely exceed 3 feet in greatest diameter and are not so well rounded as those on the Blue. On the Swan, below Galena Gulch, the gravel is more uniform and contains fewer large bowlders than that along the Blue or in French Gulch. Drilling and dredging show that the rock bottoms of the channels are smooth as a whole, but at a few places where the streams have cut into quartzite the auriferous surface of the bedrock is rough and hard enough to tax the dredge machinery severely. Shales generally are so soft as to be easily excavated by the buckets; and the porphyries, owing to more or less decomposition, also constitute a tractable bottom material.

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[a] Ransome, F. L., Geology and ore deposits of the Breckenridge district, Colorado, U. S. Geol. Survey Prof. Paper 75, 1911, 187 pp.

By no means all of the gravel is commercially auriferous. The workable strip, or "channel" in dredging parlance, is generally 180 to 400 feet wide. It follows a sinuous course along the valley, has no regular relation to the channel of the present stream, and does not invariably correspond to the deepest part of the bedrock trough. The lateral limits are indefinite and irregular; like the tenor of the deposit they are clearly affected in many places by the proximity of lateral gulches. Thus the gravels of French Gulch are especially rich below Nigger Gulch, and some of the best ground known on the Swan is at the mouth of Galena Gulch.

MONTANA DREDGING FIELDS.

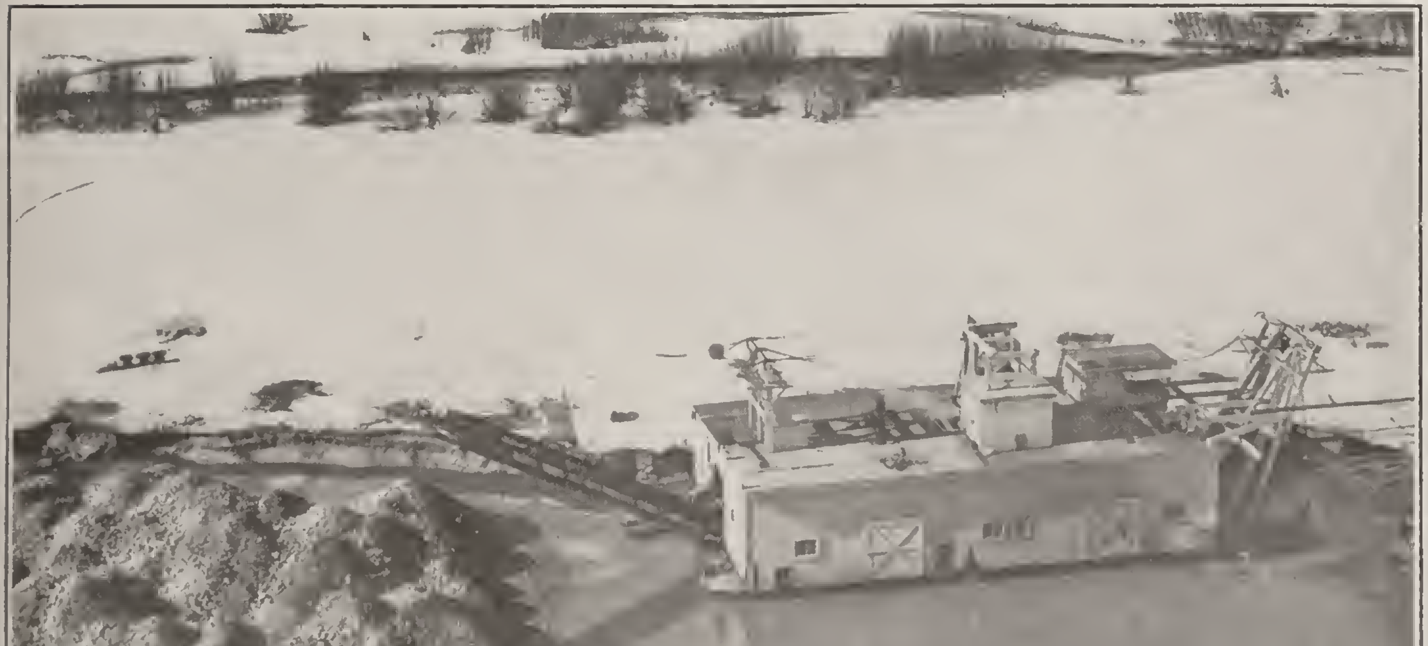
The most important dredging operations in Montana are on Alder Creek, Madison County, at Ruby. (Pl. VII, *A*.) In fact, this is the only field, except for early work at Bannack, where gold dredging has been successful in Montana.

Alder Creek rises on the northern slopes of Old Baldy Mountain, at the south end of the Tobacco Root Mountains, and flows northwest in an irregular curve to the Ruby River at Laurin. The prevailing rocks in the Tobacco Root Range are schists and gneiss, with limestone and quartzite in places. The auriferous gravels at Ruby lie on volcanic ash beds that form a smooth "false bedrock" that shows only gentle undulations and slopes regularly to the west. In the gulch itself occasional spurs of the true or original bedrock are struck in dredging and are in part responsible for some of the ensuing repairs.

Auriferous gravels extend at least 16 miles along Alder Gulch, but the area known to be profitable for dredging is probably only about six or seven miles long, and contains about 1,750 acres of proved dredging ground. Most of this ground lies at the mouth of the gulch and below, where the waters of Alder Creek have formed a low alluvial fan. The depth of the deposit increases gradually toward the west and in places at a distance of 2½ miles from the mouth of Alder Gulch is nearly 60 feet deep. In places are bowlders larger than those usually found in California dredging areas and at these places the gold content is generally higher than in the rest of the gravel.

Although not as cemented nor as hard as the hardest ground in the Oroville district or the deep gravels at Folsom, the gravel has a clay matrix that makes it rather tenacious, especially near the false bedrock. The gold grains vary in coarseness, the average size increasing up the gulch. In the dredging ground 40 to 50 per cent of the gold passes a 60-mesh screen, and 15 to 30 per cent passes a 100-mesh screen. The gold varies in fineness also, that from the dredge farthest up the gulch running about 0.836 fine and that from the lowest dredge, about 2 miles below, being about 0.873 fine.

A. DREDGE PIT NEAR RUBY, MONT.



B. BOSTON AND IDAHO DREDGE IN BOISE BASIN, IDAHO.

IDAHO AND OREGON PLACER DEPOSITS.

In Idaho the placer deposits are largely confined to the large area of deeply eroded granite in the central part of the State. Most of the placer gold has been derived from the gold-bearing quartz veins with which the districts are invariably ribbed.[a] The most important placer district in Idaho is Boise Basin (Pl. VII, *B*), where the Boston & Idaho Dredging Co. has been at work a number of years.

In this basin, the ground is light and easily handled; the gravel is about 30 feet deep; and at times a dredging face 1,300 feet wide has been carried. Before dredging started there had been extensive hydraulic mining in the vicinity and much of the material handled by the dredges is tailings from the old workings. The dredges usually work through the year, but on account of severe weather they shut down in January and February, 1915. There is a great deal of sand in the ground and it is often necessary to run the top material dry through the screens.

Other dredging operations in Idaho have been carried on at Kirtley Creek and Bohannan Bar near Salmon City, Lemhi County, and minor operations at Moose Creek, Lemhi County; Pierce City, Clearwater County; and Elk City.

In Oregon the only profitable dredging enterprise as yet developed is that of the Powder River Dredging Co., near Sumpter. The Powder River drains the richest mineral district in eastern Oregon. The gravel may be characterized as a medium-size dredging gravel, being somewhat larger than the average California gravel and containing some large bowlders. It is 12 to 25 feet deep, the average depth being about 16 feet and 300 to 1,500 feet wide. A few ledges of hard country rock are occasionally encountered. The gold content is mostly in the lower beds and on or near the soft decomposed bed rock, which is known to the dredgemen as "clay webfoot."

PLACER AREAS IN PHILIPPINE ISLANDS.

In the Philippine Islands the placer areas[b] are small as compared with those of other famous fields, but are large enough to offer very profitable working. Except for a few feet at the surface, all the ground dredged in the Philippines has been below sea level and the tide flows nearly to the head of the rivers on which the dredges are.

The gold in general is much more angular than that in placers in other countries. Usually the grains are fine and occur as small well-shaped crystals. An unusual feature along Paracale and Malaguit Rivers is the gold-bearing quartz picked up by the dredge in pieces that range in size from a pea to bowlders weighing 100 kilograms.

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[a] Bell, R. N., State Inspector of Mines, Boise, Idaho.

[b] Kane, William, Gold dredging in the Philippines: Mineral Resources of the Philippine Islands, 1913, p. 39.

The gold in the Paracale district is usually found in the layer of gravel next to the bedrock and filling the crevices and cracks of the bedrock. The gold-bearing gravel is 0.3 to 3 meters (1 to 10 feet) thick. Little or no gold is found in the sands and clays that overlie the gravel.

At Gumaos little or no black sand or pyrite is found. At Paracale and Malaguit, however, large amounts of magnetite, ilmenite, pyrite, and some metallic copper, with other accompanying minerals, are found in places. In Nueva Ecija the gold is in even finer grains than at Paracale, and exceedingly large amounts of black sand make the saving of gold difficult.

On Paracale and Malaguit the mangrove roots and soil are about 2.4 meters (8 feet) in depth. Below is a layer 3 to 6 meters (10 to 20 feet) of black sticky clay or "pug," then a thin layer of sand, and then the pay gravel, which is 0.3 to 1.5 meters (1 to 5 feet) thick. The bedrock, a decomposed granite or gneiss, is well adapted for dredging.

At Gumaos there is about 1 meter (3 feet) of soil, then 3 meters (10 feet) of yellow clay and 6 to 9 meters (20 to 30 feet) of black clay or "pug," below which is the layer of pay gravel which is 0.3 to 4.5 meters (1 to 15 feet) thick. There is very little quartz in this gravel, and none of the quartz is gold bearing like that in Paracale. The recovery of gold from ground so far worked varies from 30 to 90 centavos per cubic yard. Some patches yielded as high as 2 pesos per yard. The greatest depth dredged is 15 meters (51 feet), but the average depth is about 11 meters (35 feet).

Philippine placer ground, so far as shown by dredging, differs from that in other countries in the extremely large proportion of fine material and clay. In California only from 30 to 40 per cent of the material dredged passes through the screens, and this is generally clean sand with little clay. In the Philippines a tropical climate has caused much weathering and decomposition with resulting production of clay and soil, so that 70 to 80 per cent of the material dredged passes through the screens. Also, the material, from its being largely clay, is sticky and hard to disintegrate, resulting in loss of gold. Even the gravel bed in which the gold is found has undergone considerable decomposition to clay.

YUKON DREDGING FIELDS.

Dredging in the Yukon field, which is typical of other fields in the far north, is now being conducted on two classes of deposits, river valley and creek beds. Where thawed the gravel in the valley of the Klondike River is well adapted for dredging. The gravel is free, rounded, and clean, and averages 30 to 40 feet in depth. The bedrock is sufficiently decomposed to be dug with ease. The top burden in the thawed areas of gravel has been largely removed by stream action and presents little or no difficulty to dredging.[a]

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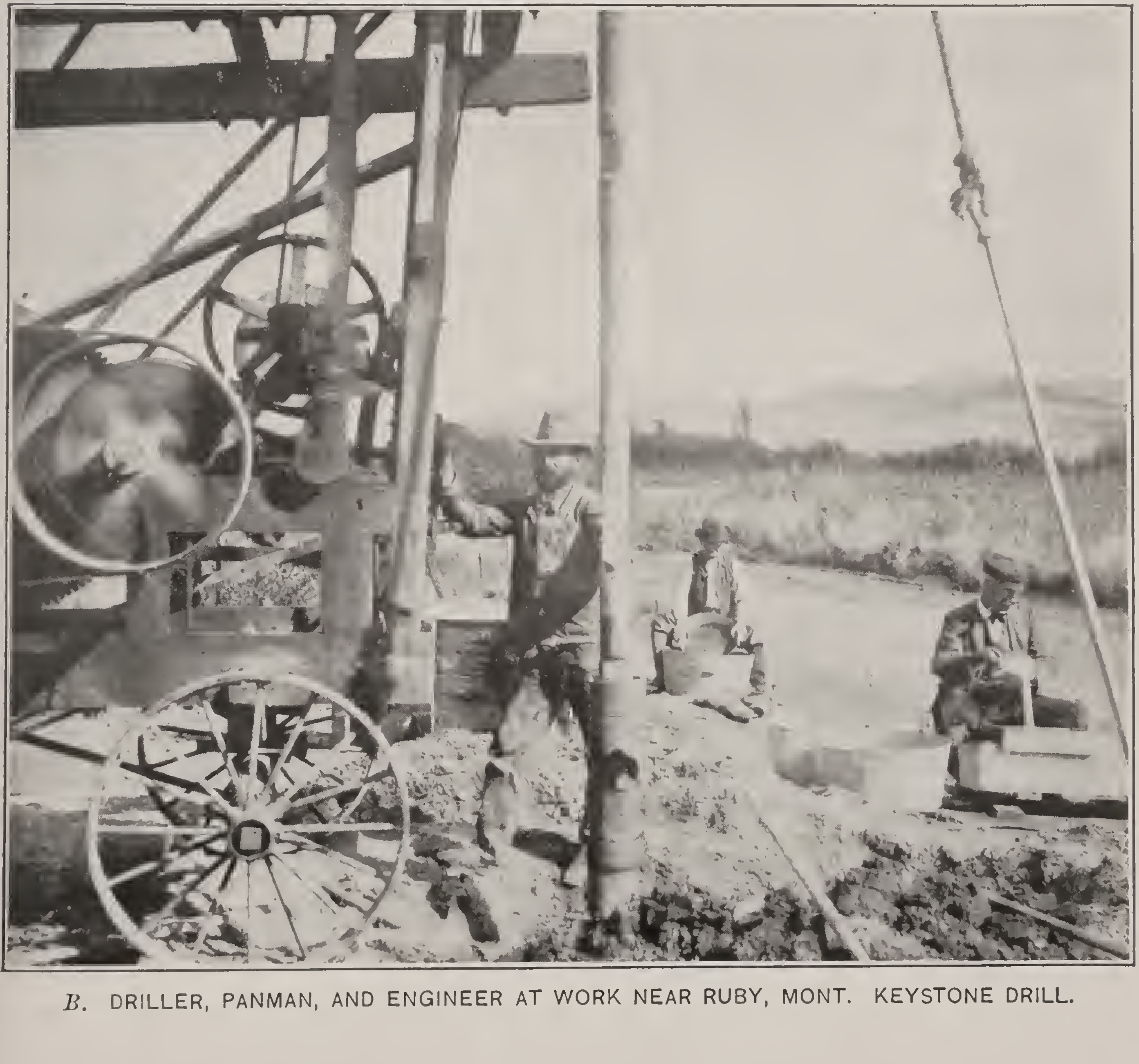
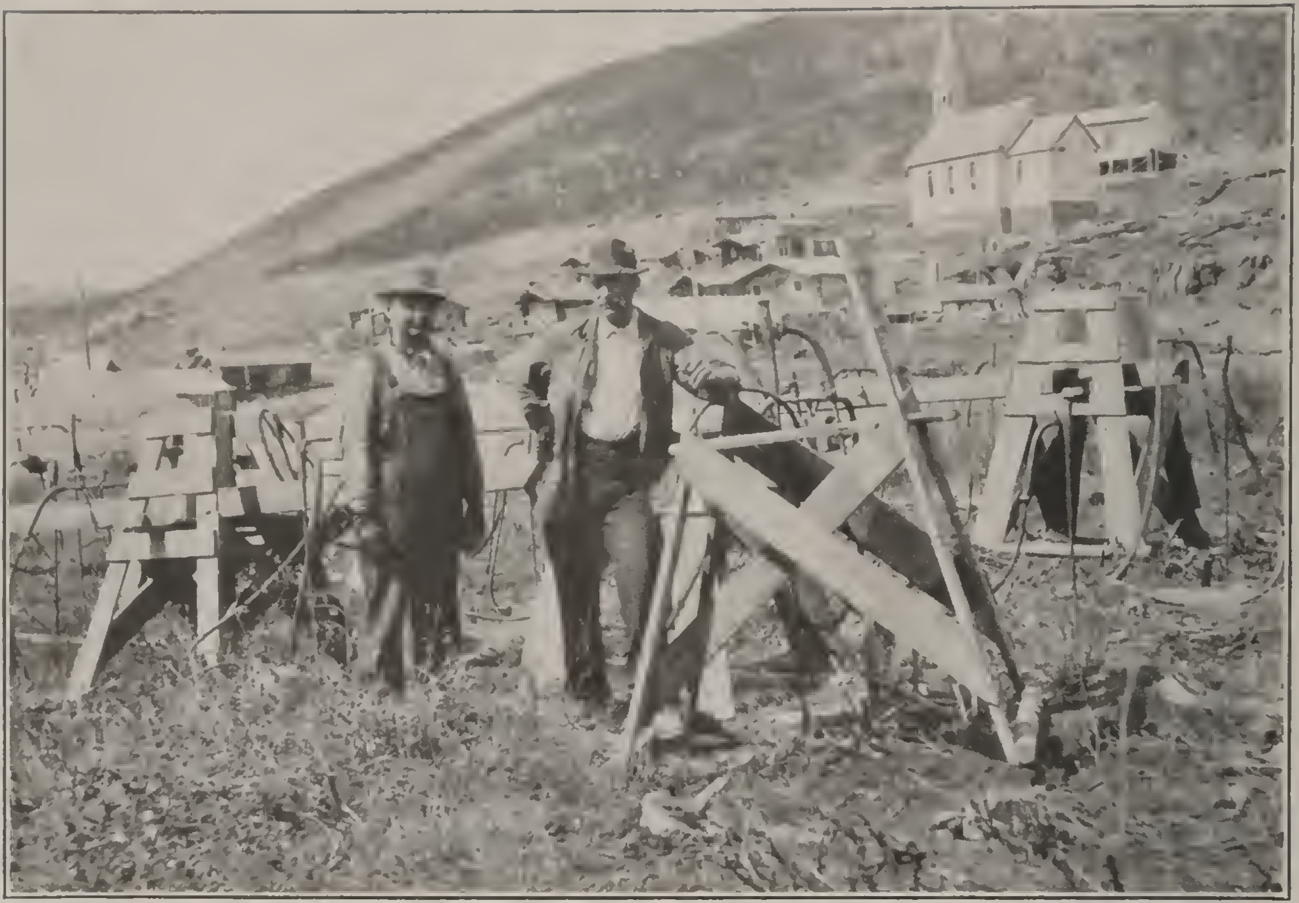
[a] Perry, O. B., Canadian Mining Inst., vol. 18, 1915, p. 42.

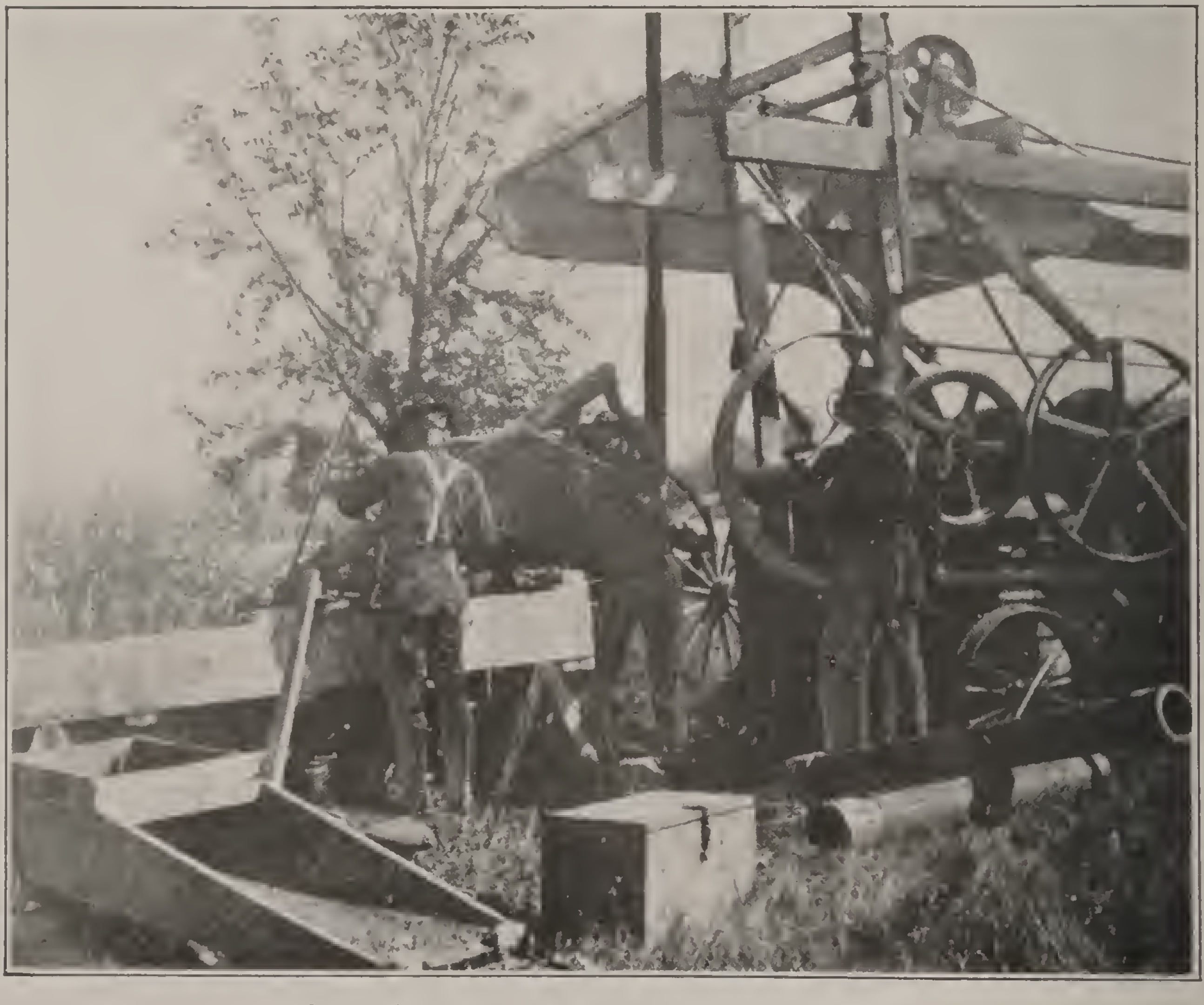
A. OVERBURDEN ON PLACER GROUND, NOME DISTRICT, ALASKA.



B. DRILLER, PANMAN, AND ENGINEER AT WORK NEAR RUBY, MONT. KEYSTONE DRILL.



A. THAWING WITH STEAM POINTS ON BONANZA CREEK, YUKON TERRITORY.



B. PROSPECTING WITH A KEYSTONE DRILL.

The creek deposits (Pl. VIII, *A*) are altogether different. They are shallow, the depths to bedrock mostly ranging about 18 to 20 feet. The gravel is angular and contains little sand, the gold is confined almost entirely to bedrock. The creek gravels are overlain with what is known locally as "muck," a deposit containing organic matter, fine sand, and water, which almost invariably is frozen, and where undisturbed is covered with a blanket of moss. The depth of this muck is 4 to 8 feet, averaging about 6 feet. In dredging on the creeks it is necessary to take up 4 to 6 feet of the bedrock, which is a massive and blocky sericite schist.

Operations in the Yukon district are subject to difficulties not met elsewhere except in Alaska. These difficulties are caused by the climate and the character of the deposits.

The extreme cold of the Northern winter limits the open season to about six months. Dredging can not begin until thawing has produced water enough to float the boats and to clear the ponds of winter ice, 5 to 7 feet thick. Work at both the start and finish of the season is hindered by cold. All moving parts, such as the tailing conveyor, must be housed and also warmed with steam coils. Also, steam must be used freely to keep the ladder and bucket line clear and to keep the gold tables in proper condition. The dredging season ends when the creek freezes, so that there is not enough water for floating the boat.

The creek deposits present many obstacles to dredging. In the first place the deposits are shallow, which necessitates frequent moving of the dredge, both "stepping ahead" and across the pond. Also the shallow ground increases the labor for necessary outside work, such as setting "deadmen" and removing stumps. The thawed muck is not difficult to handle, particularly if it is sandy, but it often contains buried stumps, roots, and other material, which tend to choke the feed hoppers and cause delays. Digging the bedrock in order to recover the gold is the most difficult part of the process. It is slow, hard work that entails heavy wear on the entire machine and necessitates continual replacement of all digging parts. Frozen ground is the greatest obstacle to dredging on a large proportion of the creeks, for the gravel must be thawed (Pl. IX, *A*) before dredging can go on successfully.

The short working season led to many departures from usual methods. Effort is concentrated on getting all that is possible out of the dredges and the result is a remarkably high efficiency for the period of operation. Each season has shown a gain in yardage per dredge hour over the season previous, until the present rate compares favorably with that of dredges of the same size in California, where conditions are much more favorable. This result is largely made possible by thorough overhauling and repair during the months of no dredging.

PROSPECTING DREDGING GROUND.[a]

To assure the success of a gold dredging venture, experience and judgment are requisite to the conduct of every detail of the work and preeminently in prospecting and examining the ground.

Many failures in dredging are traceable to lack of thorough examination of the ground before putting in a dredge. The tendency to build an expensive plant before demonstrating the value of the property was as pronounced a fault in the early days of dredging as in those of other branches of mining.

Improvements in construction and equipment and the overcoming of seemingly insurmountable difficulties have greatly broadened the dredging field and have relieved the engineer of many of the difficulties that confronted him a few years ago. A thorough investigation of a placer deposit, however, is not as simple a matter as the investor is often led to believe. Such an investigation should never be made by a man unfamiliar with the factors governing the determination of the value of placer ground and with the conditions necessary for successful dredging.

It is difficult to give any one factor as most important in the examination of dredging ground, as several conditions must be favorable to insure success. The gold value in itself may be of minor importance in valuing some ground, because a hard uneven bedrock, many bowlders, much clay, or a rough, bedrock surface contour may prevent the profitable dredging of ground with a comparatively high gold content. However, rich enough gravel, the feasibility of economic work, and enough ground to warrant the installing of at least one dredge are the salient features.

To test the value and character of the gravel and the bedrock either shafts or drill holes (Pl. X) must be sunk. Gulches, old prospect shafts, pits, or hydraulic faces may be available and give considerable information in a reconnaissance of the ground, perhaps sufficient to warrant an engineer of experience in advising his clients to make no further outlay in the examination of the property. But an engineer would not be justified in reporting favorably on the ground without making a complete examination. Besides the conditions that influence the actual operation of a dredge, such as depth and character of bedrock, possible fuel or power supply, labor conditions, etc., there are others of almost equal importance, which if unfavorable

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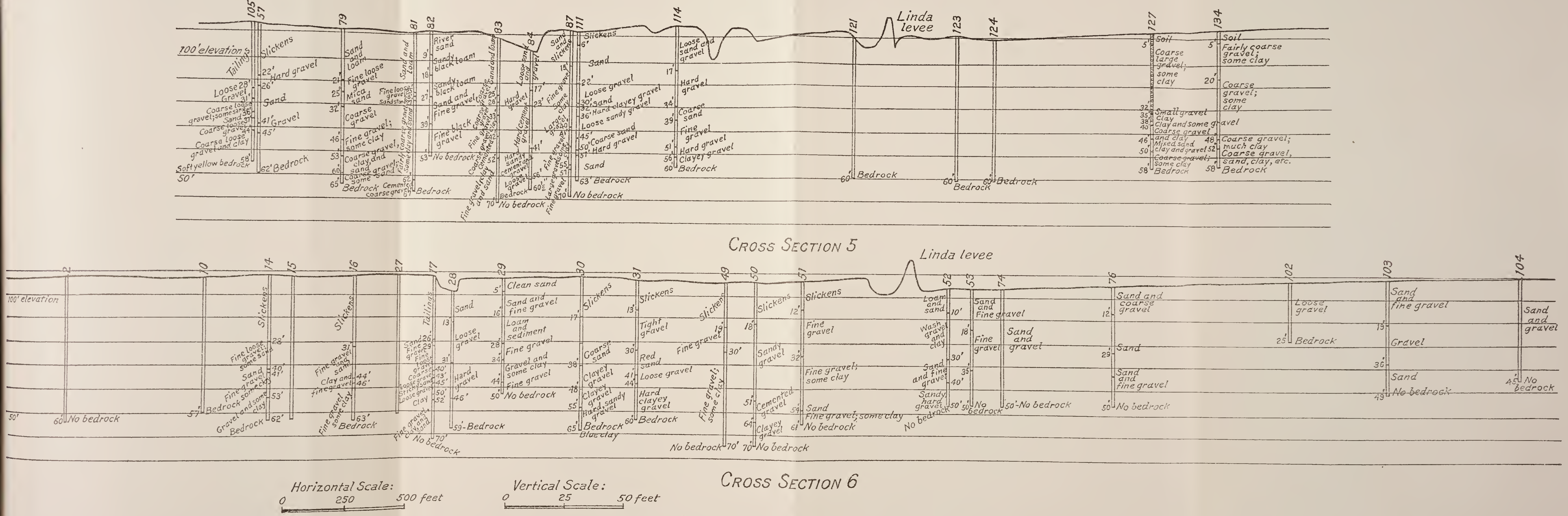
[a] In preparing this section the author has taken a chapter from "Gold Dredging in California," by W. B. Winston and Charles Janin, California State Mining Bureau, Bull. 57, 1910, as an outline. Such material as was necessary to bring the chapter up to date has been added.



CROSS SECTION 5

CROSS SECTION 6

SECTION SHOWING LOGS OF DRILL HOLES IN DREDGING GROUND.

would result in the engineer turning down that at first glance seemed meritorious. Among these factors are transportation facilities, which govern the cost of supplies and of installation, the price of the ground, titles, and climate.

FEATURES THAT MUST BE DETERMINED.

Proper examination of a placer for dredging, therefore, involves the determination of the following conditions:

1. Value, character, and distribution of gold content.
2. Depth, character, and quantity of ground to be worked.
3. Character and contour of bedrock.
4. Water level and available water supply.
5. Costs of fuel, power obtainable, possibilities for hydroelectric power.
6. Labor, transportation, supplies, etc.
7. Surface contour and timber growth.
8. Operating costs, which is influenced by all the physical factors.
9. Cost of land, royalties, titles, etc.
10. Climatic conditions.
11. In foreign countries other factors to be considered would be taxes, duties, stability of the Government, etc.

In determining the method of prospecting, shafts should always be used, if possible, in preference to drill holes, as the latter give less reliable information in regard to the physical conditions and the probable gold content of the gravel. Shaft sinking permits a much larger sample to be taken, gives better opportunity for examining the gravel, and if the material taken out is handled carefully, lessen the chance of error in the work. Shafts, however, are necessarily limited to favorable conditions, whereas drill holes can be sunk in any class of ground. It must be remembered also that the value of nearly all the dredging ground in California and elsewhere has been computed from sampling with drills and that, if other conditions are favorable, the largest dredge operators in the world would not hesitate to finance a dredging venture the mineral content of which was estimated entirely from drilling tests by engineers whom they believed reliable.

In prospecting the chances for errors are great, and constant care is necessary. A careless drill runner may neglect to drive the pipe ahead of the drill and pump an excessive quantity of material from below the shoe when pumping the drillings, as is frequently done. Careless work may also be done in drilling compact or frozen ground where casing is not thought necessary. Sometimes such carelessness accounts for a high gold content that is not found by the subsequent dredging. In loose ground, the pipe may be driven too far before pumping and become clogged so that the volume of the core drawn up does not increase in proportion to the depth of driving. In running, sandy ground far more material than that due to the length of drive may run into the casing. Allowance must be made for this, and often the casing has to be driven several feet until it reaches a firmer stratum. Generally such loose ground is barren or nearly so; if it is not, the greatest care and judgment are necessary in handling the hole and estimating the contents. Small bowlders in loose ground may cause the displacement of considerable underlying material before a bowlder is broken or pushed aside.

The risk of salted samples from either shafts or drill holes depends upon the conditions of examination, the care taken, and the experience of the engineer. Instances of salting in both shaft and drill sampling are numerous. A common precaution when one shift is employed and the hole is unguarded overnight is to put the driving blocks on the drill stem and leave it resting in the casing. As this practice is not absolutely effective, it is well to put a pan or two of barren tailings in the bottom of the hole, to be pumped out and panned the first thing in the morning. It is more easy to salt the examining crew in drill work than in shaft sampling, but with proper care there is small danger of salting being attempted without discovery. In this connection, the following paragraph is of interest:[a]

> There are two cases in the Philippines where placer deposits were tested and reported to be of value and later were found to valueless. In one case, a dredge was purchased and installed, failing almost immediately because there was not sufficien gold value recovered. Another property was reported to be very rich, and remarkable samples were submitted by the claim owners as proof. A company was formed and stock was placed on the market, but fortunately the error in valuation was discovered before much money had been spent. It appears that the samples in the two case cited above had been salted.

A number of attempts at salting in the United States and Alask where the examining engineer detected the "salt" could be cited Other cases could be given of salting and careless or inexperienced work being responsible for large losses of money spent on the propert or in building a dredge.

When because of the presence of water, or for other reasons, shaf sinking is impracticable, prospecting must be done with drills. Experienced men should be engaged, and every precaution taken t insure the results indicating as nearly as possible the gold content o the ground prospected.

The gravel may be fine and sandy, medium, or coarse. In fin gravel the pebbles are usually less than 2 inches in diameter and generally are much smaller; in medium gravel, the diameter of the pebbles range up to 6 inches; and in coarse gravel much of the materia is larger than 6 inches.

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[a] Eddingfield, F. T., Mine exploitation and the causes of some mine failures in the Philippine Mineral Resources of the Philippine Islands, 1913, p. 55.

Compact gravel, if not cemented, can be worked with a pick, and a pipe can be driven in it without much difficulty. Loose gravel is easily worked with a pick, but shafts in it must be lagged. Quicksand or fine loose gravel containing an excess of water is liable to give by drilling an excess of material for the distance sunk, and the results from shafts in such ground would probably be too high as well, if indeed shaft sinking were possible.

SIZE OF PROSPECTING SHAFTS.

Prospecting shafts should be of uniform diameter from top to bottom, in order to facilitate computation if the entire content of the shaft is to be washed, but often the practice is to cut a small section along the side of the shaft from top to bottom instead of washing all the gravel. Usually it is considered good practice to check all results obtained from putting the entire contents of the shaft through a rocker or sluice box as follows: Channel down the sides, staggering the cuts at different levels, measure the cuts carefully in all dimensions at various points, then average these dimensions, calculate the volume from the averages, and weigh the gold obtained therefrom.

Square shafts are seldom used unless the ground is too loose or wet to stand and needs timbering, then shaft sinking becomes expensive and soon passes the limit of economy. In wet ground it is also the practice to use round iron caissons, usually in 3 or 4 foot sections, in place of timber lagging, and to sample the entire content of the shaft. In very wet ground, a nest of caissons made of ½-inch steel in 4-foot sections ranging from 5 feet to 3½ feet in diameter, and tapering slightly to allow them to telescope readily, has been furnished by some manufacturers. Each section has a steel band, 2 inches by ½ inch, on the outside of the top end and on the inside of the bottom. The bands stiffen the caisson and prevent slipping of the different sections. With these caissons and a good centrifugal pump operated by a gasoline engine, shafts that would be impracticable under other conditions, have been sunk in wet ground at a reasonable cost. Care must be taken at every step of the process as in drilling, and occasionally in very wet ground a small caisson is driven ahead of the shaft, and a sample is obtained before the gravel is removed from inside the large caisson. This method, it is thought, obviates most of the danger of inflow of fines from outside.

In very wet ground, however, better results may be expected from sampling with drill holes. When a heavy flow of water is encountered in a shaft, the shaft can be discontinued and the remaining distance to bedrock can be sampled with a drill hole.

A convenient size for round shafts is 41 inches in diameter, which gives approximately 1 cubic yard of material from each 3 feet of depth. Although shaft sinking is not generally considered cheaper than drill tests, under favorable conditions the cost of shaft sinkin may be considerably less than $2 per foot, according to local wage and conditions. The cost varies greatly in different places, and tim bered shafts for prospecting gravel have cost as much as $25 a foot In one Colorado field, on the other hand, under favorable condition more than 100 shafts ranging from 5 feet to 30 feet deep were sun at an average cost for sinking, not including sampling, of 24 cent a foot. The cost of labor was $1.50 and $1.75 a day.

PROSPECTING BY SHAFT AND DRILL HOLES.

The following figures show the gold content indicated by drill hole sunk with a standard Keystone drilling rig, and the use of the usu factor, and by a shaft 3 by 3 feet sunk around the drill hole, all t material from the shaft being washed. The figures are of especi interest, as such tests are not often made.

Gold content indicated by a drill hole and a shaft.

Test No.	Content as indicated by drill sampling.	Content as indicated by shaft sampling.	Depth of ground.
	Cents per cubic yard.	*Cents per cubic yard.*	*Feet.*
0	3.8	6.5	12
2	34.0	35.0	25
3	9.4	6.5	29
4	9.2	50.0	25
5	16.0	14.4	15
6	13.7	25.5	14
7	5.0	7.0	17.5
8	13.2	12.3	15

Of the eight samples shown the shaft gave a higher gold content f five and the drill holes for three. Unquestionably shafts are far mo reliable than drill holes for prospecting placer ground where sh sinking is feasible, and on the ground sampled the reason sinking shafts was that the core obtained from drill sampling w less than the theoretical core and it was desired to check the dr hole results. Dredging on this ground has since demonstrated other parts of it, a higher gold content than that indicated by drill, but none of the ground prospected by shafts had been dredg at the time this was written.

In prospecting with drills in California, the Keystone No. 3 tract machine (Pl. IX, *B*) is generally used. This machine is self-c tained, has an 8 or 10 horsepower boiler, and a walking beam to w the tools. Wood, coal, or oil can be used for fuel, and where elect power is conveniently available the boiler can be discarded for electric motor.

The casing generally used has an inside diameter of about 6 inch is $\frac{6}{16}$ inch thick, and weighs about 28 pounds to the foot. It is into sections 5 to 7 feet long, screw threads being added as additio

pe is required. The pitch of the casing thread generally used is
out eight to the inch; but in Oroville and some other districts,
avier threads with about half this pitch are commonly used, as
ch a thread is thought to better stand pulling from deep holes.
number of extra lengths are kept for use at those times when all
e casing can not be drawn from deep holes. This practice elimi-
tes much loss of time and extra expense incident to cessation of
erations while waiting for extra pipe or digging out pipe that can not
withdrawn by usual methods. The outside diameter of the cutting
oe is about 7½ inches. The threads of the pipe and couplings are
eaned from grit, and are generally slightly lubricated with graphite;
le grease is also used but graphite is considered preferable as the
ance of getting grease in the hole is lessened. The threads of the
pes and couplings should be in good condition, to prevent loss of
pe in the hole during pulling.

Drilling without casing has been done in hard ground that will
and without the pipe, but is not good practice, and if the result
ows high values, they can not be accepted as truly indicative of
e gold content. Moreover, ground hard enough to stand drilling
ithout the casing may be too hard for the dredge to handle. In
laska it is customary to drill frozen ground without using casing,
though ground that is only partly frozen should be cased.

The casing pipe is marked with plain figures for each foot in depth-
the operator desires, some private mark may be used. The drill
pe is also marked. Records of depth and all details of operations
e usually kept by the panner. Unless the record of pipe depth is
curately kept, it becomes almost impossible at times to determine
hether the drill bit is striking above or below the cutting shoe.
r the purpose of a constant base to measure from, it is best to
ace a fairly long, wide, and thick board on the ground near the
ill hole on which the drill man can also stand while working. Also
e marking of the drill rope should be occasionally checked, as ropes
retch and bits wear with use.

The bit and stem, which weigh 800 to 1,000 pounds or more,
e allowed to drop with a slight slack in the cable, so that their
pact loosens the gravel. Tests have demonstrated the advan-
ge of quick, long strokes, 36 to 40 inches long, 56 to 60 a minute.
ith a slow stroke the loosened material is apt to settle between
okes, causing most of it to be recut and also causing possible loss
gold, aside from the loss of time. Obviously, a hole will be sunk
icker if the bit hits the clean core at each stroke, but the saving
time is a secondary consideration in sampling placer ground,
curacy of results being the prime factor.

HANDLING OF BITS.

Bits are of several types, the bit most suitable for gravel and san being thin bladed; for drilling large bowlders or into hard bedroc this bit is sometimes replaced by a heavy rock bit that has a cuttin edge of wider angle. In loose gravel such a bit would pack th material and might drive some of it from below the cutting sho to the side of the pipe. It is important that the bit be kept shar and well beaten out at the edges. Excessive drilling through th use of a dull bit may cause flouring of the gold.

USE OF SAND PUMP.

The sand pump is a hollow steel cylinder, 8 feet long and 4 inche in diameter, which has a valve at the bottom and a piston, or sucke rod, provided with a valve. The piston travels the whole lengt of the cylinder, going to the bottom of the pump when lowered, an when drawn up rapidly causing a vacuum that draws in sand, mudd water, and small stones. The efficiency of the pump depends o the rod being lifted quickly. Occasionally the valves require repai ing, and sometimes an obstruction in the bottom valve seat, whic can be readily repaired, causes leakage. A good pump will draw u everything in the casing that is loose and small enough to run i gold and other minerals are drawn in with the sludge and are hel in the pump by the foot valve. In drilling above the water lev or in ground containing little water, some water must be kept in th casing to facilitate drilling and permit pumping. In drilling fine medium gravel it is usually advisable to send the sand pump dow immediately after driving the casing, in order to prevent soft mat rial being forced out of the core through the coarser stones by th weight of the drill. Often it is possible to pump out the entire co without drilling at all. Various mechanical troubles occur duri drilling, but are hardly worth enumerating as they are easily repair by an experienced drill runner.

PROCEDURE IN DRILLING.

In starting a drill hole at the surface a shallow hole, like a po hole, is usually dug and the first section of the casing, with its sh attached, is tamped in firmly, and plumbed with a spirit lev. Care must be taken to keep the casing perpendicular, otherwise t may become bent and the hole lost before bedrock is reached; also bent pipe is difficult to pull. It is usually advisable for the help to climb the scaffold and steady the tools (bit, stem, and socke) with his hands for the first few feet of drilling, so that the tools w not hit to one side of the center and knock the casing out of plum.

If the driving shoe strikes the sloping side of a hard bowlder, t casing may bend and cause the loss of the hole, though in fai loose material the work may be continued by turning the pipe slightly in the hole.

If the hole is started in top soil, the casing generally is driven until firm ground is reached, and in hard gravel far enough to remain in place while drilling goes on. In loose gravel some men drive the pipe 10 or even 15 feet before beginning to drill. This procedure, though permissible in places where the upper gravel is known to carry little or no gold, is dangerous where the depth to the mineral content is not known; but the practice is often encouraged by the drill runner, as it makes time. After the pipe has been driven the desired depth the core is loosened by the drill for a depth of about one foot and then pumped out. This process is repeated until all the core, except a few inches of the bottom, has been removed. Some operators drill several feet before pumping, as greater speed is gained in this way, but pumping is generally done for each foot or less drilled, especially when drilling in the "pay streak," where several pumpings are necessary to clean out the drilled core. In drilling where the location of the pay gravel is known and the material above the pay is thick and barren, it is the general custom to drill 3 to 5 feet for each pumping while going through this overburden.

HANDLING OF DRILL CORE.

During the drilling of a hole a core sufficient to keep outside material from entering should be left in the pipe before and after every pumping. The length of this core varies with the nature of the ground, but generally is 2 to 6 inches; it should be long enough to prevent any inrush of water or material from the sides. Water s also kept in the drive pipe to offset the pressure from the outside, as well as to facilitate drilling. When drilling cemented gravel or when the drive shoe encounters hard bowlders it sometimes becomes necessary to drill below the casing. In such an event the pipe hould be driven after drilling and before pumping whenever possible. Occasionally it is necessary to pump material from below the drive hoe. Any increase in the gold colors panned at such times must be specially noted and the quantity of material must be compared with that recovered under normal conditions. Some operators advocate measuring the material from each foot drilled, and while his would be good practice in drilling below the shoe the usual ustom is to measure all the material at the end of the work at each hole.

The gold content of a placer is seldom evenly distributed, but is sually concentrated near the bottom, one-tenth of the material rilled often containing most of the gold; at times gold is found in ifferent layers throughout the placer, as described in the chapter on geology, and the greatest care in drilling the rest of the ground becomes useless if an error is made while the pay streak is being drilled.

Any material that adheres to the drilling tools as they are raised from the hole is washed back into the casing, so that it may be brought up in the sand pump. The drill runner keeps a bucket of water at hand for this purpose. Two or three pumpings are generally sufficient for each foot drilled, but the number depends on the material encountered; from some holes the pump will bring up less than one-half a pan at a pumping and from others even two pans. The sand pump is used until all the drilled material is obtained. If the drill has been working below the casing excess material is likely to be obtained by repeated pumpings and a careless drill runner has often "salted" himself in this way. The contents of the sand pump are emptied into a sample box made either of wood or galvanized iron, the solid material generally being caught in a gold pan placed inside the box. The contents of the pan are washed and any gold colors counted and recorded. The tailings from the gold colors are generally segregated in sizes according to an arbitrary classification and recorded on the drill log under the headings, one, two, or three; bunches of small colors representing one or more colors of the various sizes are grouped by an experienced panner.

The following figures from the drill logs of one examination show the total number of colors, counted, under the headings, for three holes. Two of these holes were drilled within a few feet of the third in an attempt to check the results first obtained. The depth drilled in each hole was about 60 feet.

Arbitrary classification.

Hole No.	Size of colors.			Total weight of gold.	Gold content of ground.
	1	2	3		
				Mg.	*Cts. per cu. yd.*
A	14	79	117	725	72.4
B	5	39	338	327	32.3
C	3	8	342	125	12.5

Weighing tests are made during the first stages of the examination to determine the weight of gold colors and to enable the panner to become familiar with some of the characteristics of the gold. On extended examinations good panners become able to approximate closely the gold content of the sample when panning the material from the drill hole.

Gold colors vary greatly in different districts and a man familiar with one district may be misled at first when panning in a new field. On one examination the gold colors weighed 200 to a cent and on another 1,200 colors were required for the same weight. In different fields the colors from drilling may vary from such sizes to those weighing a milligram or more. In Alaska and in northern California colors weighing several milligrams are occasionally brought up by the sand pump.

PANNING.

Panning is done over a tub and the tailings from the pan are afterwards rocked by the panner, as is also any material left in the sample box when enough has accumulated or when the panner has time. A simple method of handling the cores is to place the rocker on its proper grade at the overflow end of the sample box. If an apron is used in the rocker and it is occasionally shaken a little to prevent caking, very little gold will get away with the overflow. Then, after the hole is down, or at any convenient time, the panner can take the residue from the tub, put it back in the sample box, measure it, and then run the whole through the rocker. In rocking care should be taken to prevent caking at the lower end by stopping occasionally and scooping the material from the lower end back into the rocker screen. The tailing from the rocker is tested by careful rerocking or panning in order to check the washing of the sample.

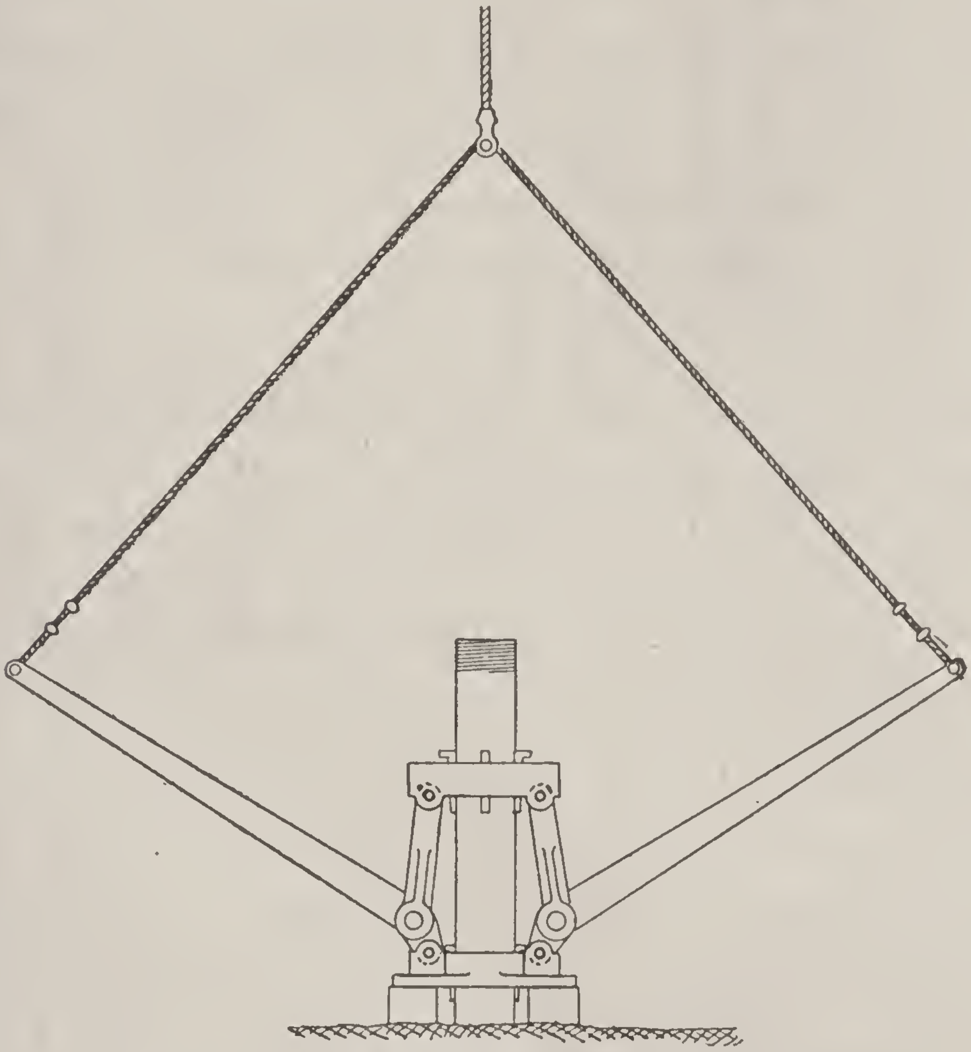

FIGURE 1.—Patented device for pulling drill-hole pipe.

After the core drilled has been removed by the sand pump, the casing is again driven and all operations continued as before until the hole is down to bedrock. When bedrock is reached, repeated use of the sand pump is necessary to remove all the drillings from the hole, and sometimes when the bedrock is soft the pump loosens material that has not been drilled. The color of the drillings from bedrock is generally noticeable and readily recognized. If the colors recovered appear excessive, the drill hole may have cut a rich seam or a small pocket and the result should be noted separately from the record of the rest of the hole.

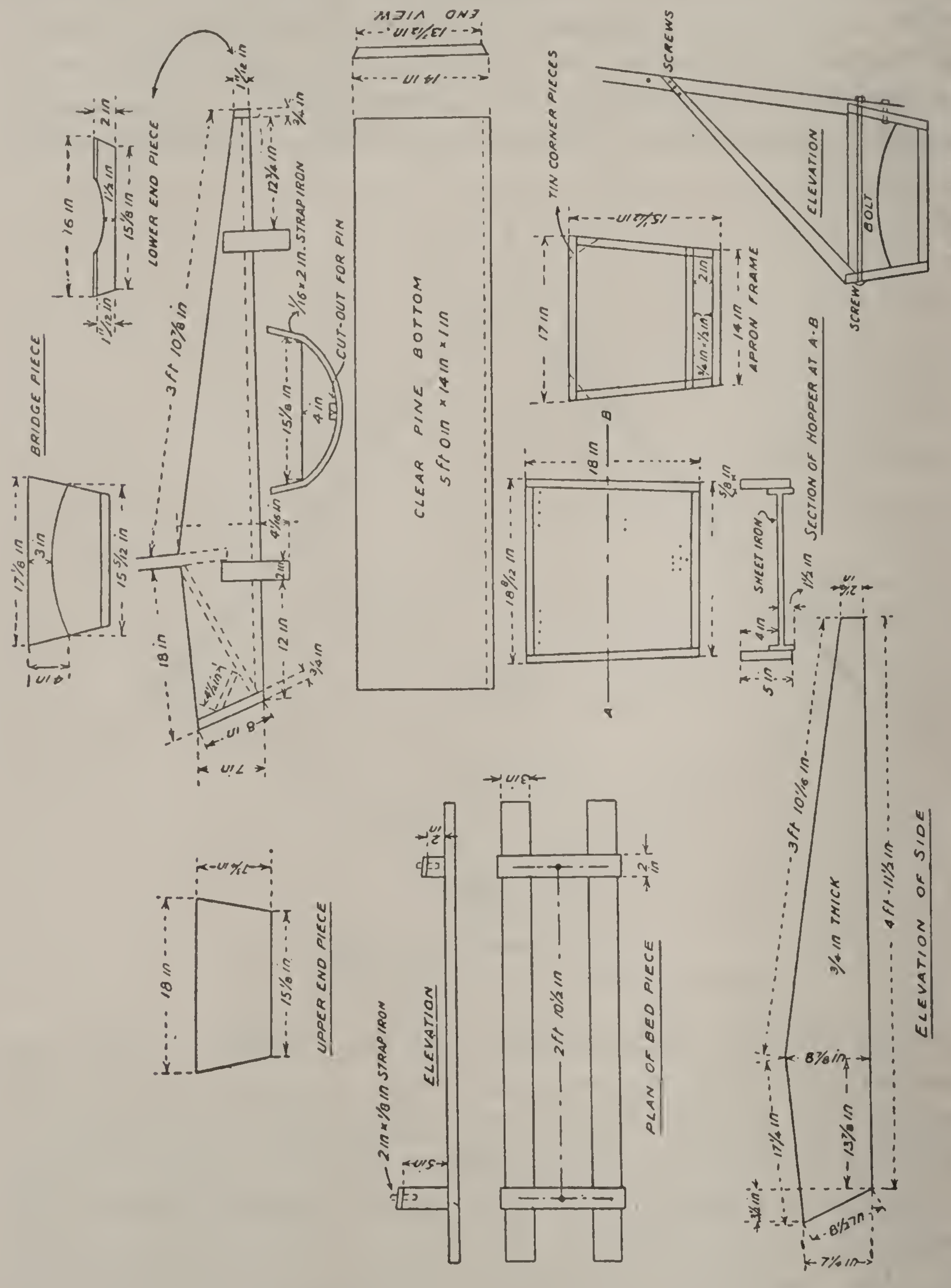

FIGURE 2.—Details of rocker.

The hole being finished, the next operation is pulling the pipe. Sometimes considerable difficulty is experienced in pulling the casing from a deep hole. A pulling jar and pulling head are generally all the special equipment necessary when a hole is shallow; with a deep hole the jars may not be able to loosen the pipe and two jackscrews may be required. A patented pulling appliance that seems to have considerable merit is shown in figure 1.

Boxes for the pumpings are made in different sizes; some are 12 by 14 inches by 8 feet long. A convenient box is made of 2-inch planks, it measures 20 by 20 inches and is 5 feet. Such a box held together with rods, is strong and easily transported. A galvanized-iron box is more easily cleaned and transported than a wooden box and will be found decidedly satisfactory on extended examinations.

The size of the rocker varies with the personal ideas of the user and is often governed by the size of the material obtainable. A good size is 18 or 20 inches wide by 4½ or 5 feet long (see fig. 2) and is used either with or without an apron. The use of the apron sometimes facilitates rapid work, because the tailings are not rerocked except occasionally to test the rocker efficiency. In the plain rocker the tailings are usually rerocked one or more times, according to the gold content of the ground. Clean water is always preferable for both panning and rocking. Some engineers line the bottom of the rocker with burlap. The writer has made tests with burlap, corduroy, canvas, and blankets and considers ordinary burlap, which is always obtainable, the most satisfactory. Some firms supply a steel rocker that probably has some advantages. For examinations, in districts where suitable lumber for a rocker might not be available experienced engineers generally carry a rocker that can be taken apart and compactly packed. A steel rocker might answer the purpose equally well.

The tubs in which the panning is done are either galvanized-iron washtubs or half barrels. The fine colors from the pannings and rockings are usually kept in a small dish and are amalgamated at the completion of the work. The amalgam is put in a small vial which is marked with the number of the hole, and the quicksilver is later separated from the gold by first using diluted and then strong nitric acid, after which the gold is washed, dried, and weighed. Some operators use no quicksilver and separate the black sand from the gold by close panning and a magnet, then dry the material, remove any remaining black sand by blowing, and weigh the gold.

If the gold does not readily amalgamate, the fact should be noted in the log book and the cause ascertained. Poor amalgamation may be due to a coating of iron oxide, the presence of arsenic or grease, or to some other cause. A common cause is the fouling of the quicksilver by grease and other impurities; these should always be kept away from the pans; but grease from the pipe couplings and from the foot valve of the sand pump will often get in. Often a little caustic potash will solve the difficulty of amalgamation, or even common salt kept in solution on top of the mercury. Sometimes so-called rusty gold will amalgamate after being rubbed with the finger against the pan. Although it is often necessary in cold weather to use warm water to facilitate amalgamation, it is best to amalgamate at as nearly as possible the same temperature as will prevail on the dredge. Methods of saving the gold in sampling that can not be duplicated or equaled on the dredge may lead to error in calculating the value of the ground. The physical characteristics of the gold determine in a great measure the percentage of recovery on the gold-saving tables of the dredge. If much light, thin, scaly, and flour gold is encountered in prospecting, the fact should be carefully noted, as there may be difficulty in saving such gold on a dredge handling large quantities of sand and gravel.

The black sand caught in the pan and riffles with the gold seldom contains anything of value. However, there is a possibility of finding platinum and various minerals of high specific gravity, and in a new district it is always advisable to investigate the black sand carefully. For this purpose a good microscope can be used to advantage.

The practice among engineers inexperienced in placer work of using the fire assay to determine gold content is rarely followed these days. A few years ago the practice was more widespread than would be imagined; numerous reports upon new districts have been based upon such methods which, needless to say, were worthless.

PROSPECTING RECORDS.

In prospecting with drills and shafts a systematic set of logs must be kept for calculations and tabulations.

It is surprising how much expensive work has been done in preliminary investigations without a systematic record being kept for future reference. Especially has the writer found this to be true of Alaskan fields; often a note of the depth and a rough estimate of the total amount of gold recovered from a drill hole or shaft that has been sunk at a cost of perhaps $100 or more are all the information available of work done, or else the whole matter was left to the memory of the drill runner. Such results might be of some interest, but would prove of little value in estimating the gold content of the ground. The logs should give all the necessary information in as simple and comprehensive a manner as possible. Some of the largest dredging companies in California have used for a number of years field, time, and prospecting logs similar to those following and find that they answer every purpose.

Form of drill log used by large California dredging company.[a]

Date ....

Hole No. .... Crew No. .... Tract ....

Started hole .... Finished ....

Depths.			Cores.			Number of pans.	Colors.			Firmness.		Lumps (clay).	Slime.	Formation.
Depth represented.	Depth of pipe.	Depth drilled below pipe.	Core after driving.	Core for depth driven.	Core after pumping.		1	2	3	Driving.	Drilling.			
......	......	......	......	......	......	......	......	......	......	......	......	......	......	....
......	......	......	......	......	......	......	......	......	......	......	......	......	......	....
......	......	......	......	......	......	......	......	......	......	......	......	......	......	....
......	......	......	......	......	......	......	......	......	......	......	......	......	......	....

Total depth ....

Depth to bedrock ....

Depth to water level ....

Measured volume ....

.... Panman.

Form of time log used by large California dredging company.[b]

Date ....

Hole No. .... Crew No. .... Tract ....

Started hole .... Finished ....

Time of day particular operation is finished.	Nature of work.	Hours and minutes consumed in—						Depth of pipe.	Remarks.
		Drilling operations.	Oiling—putting on pipe.	Pulling pipe.	Moving.	Setting up.	Delays.		
....	....	......	......	......	......	......	......	......	....
....	....	......	......	......	......	......	......	......	....
....	....	......	......	......	......	......	......	......	....
....	....	......	......	......	......	......	......	......	....

.... Panman.

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[a] Form used by company is 5½ by 9 inches and has space for 24 entries instead of 4.

[b] Form used by company is 5½ by 9 inches and has space for 30 entries instead of 4.

Form of prospecting report used by large California dredging company.[a]

Examination .... Date ....

No. of hole .... No. on map .... Crew No. ....
Location of hole ....
Method of prospecting by .... Machine used ....
Started hole .... Finished hole .... Days worked ....
Depth of top soil ....
Depth to bedrock ....
Depth drilled ....
Depth used in calculations ....
Depth to water level ....
Distribution of gold ....
Nature of top soil ....
Nature of gravel ....
Nature of gold-bearing strata ....
Nature of bedrock ....
Time consumed in: Moving .... Setting up ...... Drilling operations ......
Pulling pipe ...... Delays .... Total ....
Causes of delays ....
....
Mg. of gold recovered ....
Factors adopted .... Fineness of gold ....
Cu. ft. per ft. depth of hole .... Average value per cubic yard ....
Remarks ....
....
....
....

Signed ....
Checked by ....
Approved by ....

The field and time logs, which are kept in the field, usually by the panner or the men in charge of the drill, are made in convenient size books, about 4½ inches wide by 7 inches long, containing 50 leaves each. The paper is tough, strong, and not easily torn when wet; the books can be carried in the pocket and readily used in stormy weather. It facilitates matters to have the leaves of distinctive colors, those of the field log being usually green, and those of the time log yellow or whatever color suits the fancy of the man in charge. These books are now obtainable from technical book shops. The leaves of the field log are perforated; they are torn out at the completion of each hole and the information is copied into the prospecting log book. In the columns giving the sizes of the colors, in the field log, the largest are usually marked under No. 1 size, and the smallest under No. 3, but this method of grading the colors can, of course, be changed to suit any desired arrangement. The columns for depth and core indicate depth after pumping and the length of the core left in the drill hole after pumping. These figures added give the total depth of pipe, except where an inrush of material would cause an excess of core, which would be noted under "remarks."

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[a] Form used by company is 8½ by 10½ inches.

In the time log is kept a record of all work done and of all difficulties and delays. The record is useful for estimating the time required to prospect similar ground. For promotors and investors who are anxious for speed in prospecting, a record of this kind would be an education.

The prospecting sheet or log is also made with perforated leaves and should give a summary of the field and time log, as well as all other necessary information pertaining to drill holes or shafts.

SPEED OF DRILLING.

No average speed can be given for test drilling. Climatic and other conditions vary greatly in different localities, and an unforeseen accident to the drill or equipment might delay operations considerably. Speed should be subservient to accuracy in prospecting. However, some idea of average speed can be gained from the time required for actual tests. At one place as much as 20 feet was drilled in 12 hours in loose gravel, whereas only 2 feet was drilled in the same time in cemented gravel. On shallow ground, the frequent moving and resetting cause considerable delay, the amount depending on the distances between the holes and the difficulty in moving.

The speed of drilling generally decreases with depth. More time is required to raise and lower the tools, the casing is driven more slowly because of frictional resistance, and the casing is harder to pull. Sometimes it takes more time to pull the casing than to drill the hole, and occasionally it is economical to leave casing that does not pull easily.

On one property, the average depth drilled, including delays caused by bad weather, moving, etc., over a total period of thirty-three 10-hour days, was approximately 12.6 feet per day. The holes averaged 24.9 feet deep, the character of the deposit was medium coarse gravel, free from clay and overlain by hydraulic tailings. As the wages of the drill crew were $12.25 per day, the labor cost was seemingly $1.01 per foot drilled. In another examination three 70-foot holes required eight 24-hour days, including pulling pipe, moving, etc., and the cost for labor was $1.71 per foot. In one examination 2,500 feet of holes was drilled; the total cost of drilling and sampling on the property, exclusive of the salary of the man in charge, approximated $1.30 a foot.

COST OF DRILLING.

The following figures are from drilling in a rather loose deposit of sand and gravel, the bowlders of granite being easily broken by the drill. Twenty-four holes of an average depth of 23¼ feet, or a total of

558 feet, were sunk in twenty-six 9-hour shifts, including all delays. The cost of work on the property was a little less than $1 a foot; but as some of the men were brought from a distance their wages and expenses while traveling made the cost of the examination slightly higher.

The drill crew consists of one drill man or drill runner, one fireman or helper, one waterman and team, and one panner. When more than one shift is used, one waterman is usually able to supply the drill with fuel and water and to help in moving the drill and tools from one hole to another. The panner generally works during the daytime only, his place being taken during the night shift by a watchman, who keeps the time log, while the night runner takes care of the pannings, placing the pans in a tub in regular order, so that in the morning the panner by reference to the log can determine just where the pans belong. On some examinations where good artificial light is available and in the far north with the long period of daylight hours, two or three full crews are employed and sampling is continuous.

The daily wages of the drill crew are usually as follows:

Drill man	$4.00 to $5.00
Helper	3.00 to 3.50
Panner (white)	4.00 to 7.00
Panner (Chinese)	2.00 to 3.00
Waterman and team	4.00 to 5.00

Other expenses beside the wages of the drill crew include fuel, repairs, maintenance, and hire or purchase of a drill. In the winter months heavy rains increase the cost of transportation, making moving difficult, and greatly delaying work. A serious accident to the drill machinery may also delay the examination for several days and increase the cost considerably. In California the cost of drilling is generally considered to be $1.50 to $2.50 a foot under favorable conditions, and these figures can be used as approximations for other fields.

Too much stress can not be laid upon the care necessary to assure the samples representing as nearly as possible the value per cubic yard of each place drilled. If the work has been properly done in every detail, an estimate of the gold content of the property can be made with some degree of accuracy; if, on the other hand, the work has been carelessly done by unreliable or inexperienced men the results obtained are worthless.

PLACING OF PROSPECT HOLES.

It is generally conceded that careful washing of the material sampled will equal or exceed the recovery by the gold-saving tables of the dredge; hence, if the work has been properly done each hole may be taken as indicative of the value of that part of the ground in which it is sunk. The placing of holes is important, for even though the mechanical details of sampling each hole be most carefully done, the average values obtained may be far from representing the gold content of an area under consideration. Too many holes sunk in the richer ground would make the indicated average content too high.

In preliminary examinations (Pl. VIII, *B*) and in prospecting large areas, a few holes are first sunk to determine the probable value and extent of the workable ground. The preliminary holes should indicate whether the gold occurs in wide or narrow channels or whether its distribution is fairly even. A common method in preliminary prospecting is to choose what appears to be the most promising ground—possibly by contour features or by old workings—and to sink a row of holes across the channel, if one is found. Then another such spot is chosen and another row of holes put across. If the gold content is sufficient, an intermediate row is sunk; and if the results justify the expense of a thorough examination, the ground is laid out in regular divisions and prospected systematically. For careful prospecting the holes are not drilled in squares, but are staggered. The ground under examination is then divided into sections or squares and the holes are proportioned to each section. Where sampling has disclosed the existence of a channel, or narrow belt having a higher mineral content than the area as a whole, the outlines of the channel or belt should be determined as closely as seems warranted.

The points for taking samples can be determined only after a study of the ground. There is no rule as to the number of holes to be sunk, but ground that shows an uneven distribution of values obviously demands more tests than grounds in which the distribution is fairly uniform. Great economy of time is possible when the work is carefully planned and the sampling crew consists of men well trained and well directed. In large tracts, where the time for the examination is limited, tests to confirm samples taken in preliminary prospecting may be concentrated on an area sufficient to determine the advisability of purchasing the property and of installing at least one dredge.

VOLUME OF DRILLINGS.

The quantity of material recovered per linear foot drilled varies greatly with different material. In drilling compact gravel, the volume of core will be greater than where a number of hard, coarse pebbles are encountered. In drilling sand or fine gravel containing much water, an excess of material may be pumped, or if the casing is driven too far ahead of the drill, the pipe may become clogged and too little material be obtained. Some operators use a measuring box holding 1 cubic foot, and measure the material as it goes to the rocker, calculating that 20 cubic feet of material so measured represents 1 cubic yard actually drilled. This factor was obtained from results of numerous experiments in one field and can not be applied as a general rule. Other engineers have found that generally the quantity of material obtained exceeded the theoretical quantity by a larger margin.

Assumptions that the expansion of gravel on being taken out of the hole is very large are unwarranted; the material crushed by the drill is poured from the pan and laid down under water, under conditions closely similar to those under which it was laid down originally.

In some fields the actual recovery of material from some drill holes, in compact gravel, has been 95 per cent of the theoretical quantity; in running ground the recovery is often in excess of the theoretical quantity of core drilled. Good practice is to measure the total quantity of material pumped from each hole, and to compare the results from all holes drilled in similar material; thus, a fair average can be obtained and used as a check. As previously mentioned, greater care is necessary with material containing the mineral content than in barren ground.

A simple method of making the measurement of the total amount of material is to dig at the end of the rocker a hole about 3 feet square, and of sufficient depth to hold all material from the hole to be drilled. All the tailing from the rocker may be run into this hole and allowed to settle; the cubic contents can then be determined and tabulated.

In drilling dry ground containing small gravel with considerable sand the material recovered will be closer to the theoretical quantity than in drilling in other kinds of gravel. The theoretical quantity that should be recovered in drill tests is represented by the contents of a cylinder having a base the diameter of the cutting shoe and a length equal to the depth drilled.

DETERMINING GOLD CONTENT OF GROUND.

The area of a circle the size of the cutting shoe ordinarily used with the power drill is about 0.3 square foot, so for every linear foot drilled there would be excavated 0.3 cubic foot, or 0.011 cubic yard. As the cutting shoe soon becomes worn, the figure used is 0.01 cubic yard, or 1 cubic yard for each 100 feet drilled. Using this factor, the method of determining values is as follows:

FACTORS USED.

The value of gold recovered from a drill hole is determined in cents In a new district, the fineness of the gold must be first determined or assumed. The value in cents is divided by the number of feet drilled and the result multiplied by 100, which gives the value per cubic yard. A less simple method is to use the factor 0.27, multiplying the depth drilled by 0.27 and dividing the figure for the value of the recovered gold in cents by the product, the result being the value per cubic foot. To get the value per cubic yard the result is multiplied by 27.

One operator, who concedes the correctness of the results obtained by the use of these factors for calculating the results of drilling compact gravel, believes that different factors are necessary when holes are sunk in other material; he uses the following factors as numerous tests have demonstrated to his own satisfaction that they are correct:

Factors for computing value of drill-hole material.

(*a*) For compact gravel	0.01
(*b*) For medium gravel	.011
(*c*) For loose gravel	.012
(*d*) For loose gravel, and sand with much water	.013

In using these factors, the number of feet drilled is multiplied by the factor selected and the product is divided by the value in cents of the gold in the sample, the result being the value per cubic yard. The selection of the correct factor is highly important, and demands much experience.

If factor *a* of the method last mentioned be taken, the results of using any one of the three methods will be the same, as is shown by making a calculation for a hole 40 feet deep that returns gold to the value of 15 cents.

First method:

$$(15 \div 40) \times 100 = 0.375 \times 100 = 37.5 \text{ cents per cubic yard.}$$

Second method:

$$15 \div (40 \times 0.27) \times 27 = (15 \div 10.80) \times 27 = 1.389 \times 27 = 37.5 \text{ cents.}$$

Third method:

$$15 \div (40 \times 0.01) = 15 \div 0.40 = 37.5 \text{ cents per cubic yard.}$$

However, if the factor *c* be used, the result is:

$$15 \div (40 \times 0.012)\ 15 \div 0.48 = 31.2 \text{ cents per cubic yard.}$$

Some engineers in calculating the results of sampling by drills employ a factor different from the one in general use. They reduce the figure for the gold content of the place sampled to a value that their experience indicates more nearly represents the recovery. This practice, practically a combination of drill-hole factor and recovery estimate, involves a discount from the contents that would otherwise be assumed from the regular drill factor. Engineers following such a practice should mention it clearly in their report. Failure to do so might cause the future of a property to be further discounted by parties to whom the report was presented if they were not conversant with the methods of computation used.

One large operating company uses the factor 33⅓ instead of 27 in these calculations, the formula for the example mentioned above being:

$$15 \div (40 \times 0.333) \times 27 = 15 \div 13.32 \times 27 = 30.6 \text{ cents per cubic yard.}$$

As will be seen, the use of this factor discounts the gold content assumed for the drill hole by nearly 20 per cent as compared with the use of the factor 27.

To estimate the gold content of the area examined, the value per cubic yard as found above is multiplied by the depth of the hole in feet. The first step is to see that no erratic values for such holes are included in the general average, as the use of one such high value might vitiate the total result. A rich sample indicates a rich spot, but does not show its extent. The general practice is to reduce erratic values to a value more nearly in keeping with the general average. In this connection, Eddingfield[a] says:

> Several years ago an enginee put down 9 holes to test a certain tract of placer ground. On hole showed almost 21 pesos pe cubic meter (16 pesos per cubi yard); the other 8 holes eac showed only 1.30 pesos per cubi meter (1 peso per cubic yard) The engineer averaged the 9 hole and by doing so obtained a valu almost three times as large as i would have been had the on high value been lacking. Th error of such a method is apparent.

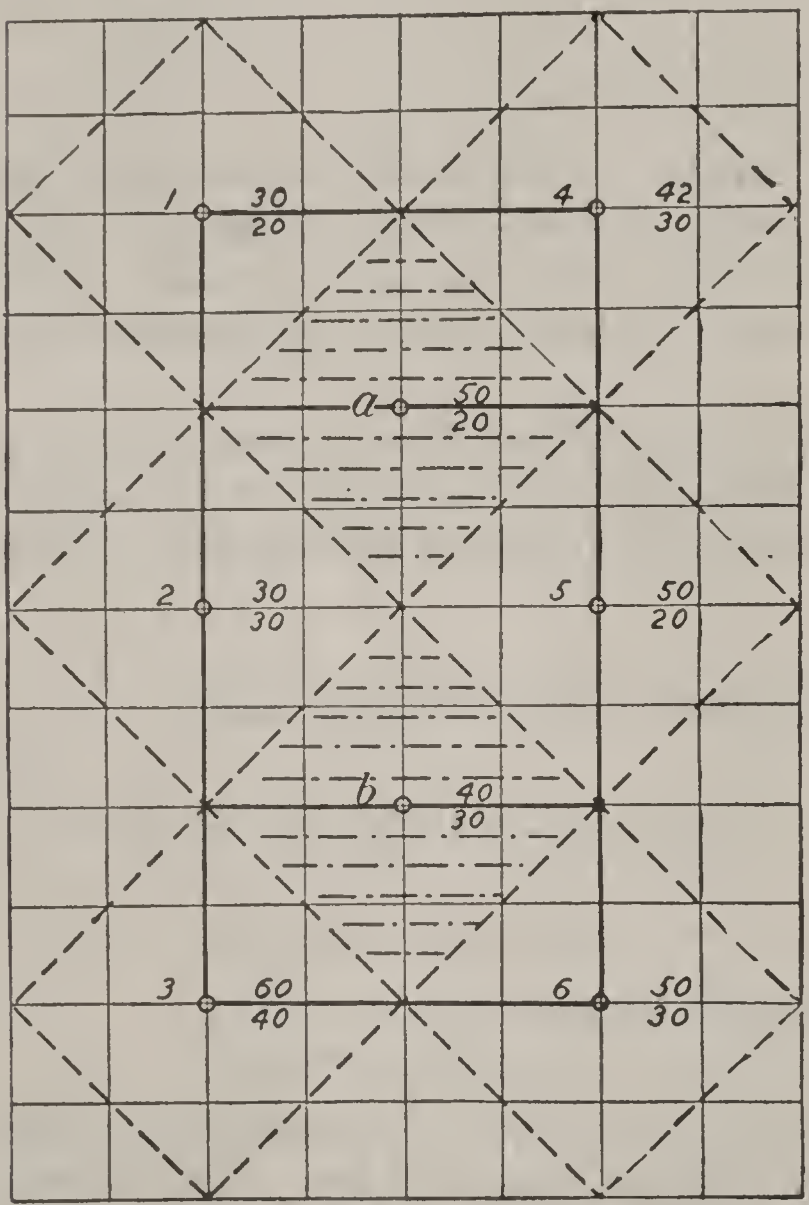

FIGURE 3.—Diagram showing weight to be given results from different holes.

After the reduction c any high values, the sur of the figures for all hole in cents per foot, is divide by the sum, in feet, of th depths of all holes, provided the drill holes hav been equally spaced an each represents practically the same amount of ground. The mai difficulty in this calculation is to select the area represented by eac hole. If we assume that the given property can be divided into number of squares of equal size and a drill hole put in the center each square then the areas represented by the individual drill hol will be of the same size. This condition is rarely found in practic and the following example of an extreme condition is given simply illustrate the method of calculation though in practice it would unlikely that the holes would be so inefficiently placed. The calcul tions are made to cover the area inclosed within the heavy lines shown in figure 3, though the drill holes could properly represent the area covered by the eight squares.

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[a] Eddingfield, F. T., Mine exploitation and the causes of some mine failures in the Philippines: Mine Resources of the Philippine Islands, 1913, p. 54.

Holes *a* and *b* would represent the shaded squares, and holes 1 to 6, while representing the same area as *a* and *b*, could be used only with a weight proportional to the part of their respective square that is inclosed within the heavy lines.

Data used in calculating gold content of area shown in figure 3.

Hole No.	Value of gold content, cents per cubic yard.	Depth, feet.	Proportional area.	Volume (column 3 x column 4).	Total foot cents.
1	2	3	4	5	6
1	30	20	0.25	5	150
2	30	30	.50	15	450
3	60	40	.25	10	600
a	50	20	1	20	1,000
b	40	30	1	30	1,200
4	42	30	.25	7.50	315
5	50	20	.50	10	500
6	50	20	.25	7.5	375
Total				105.0	4,590

$$\frac{4590}{105}=43.7 \text{ cents per cubic yard.}$$

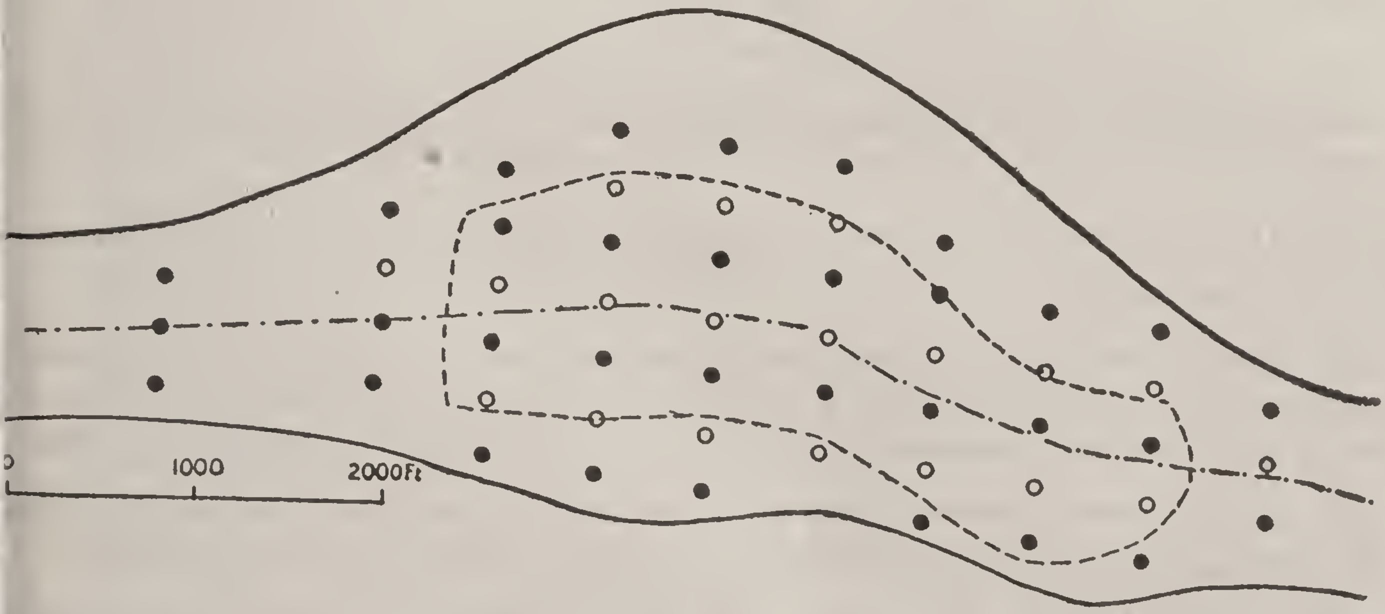


FIGURE 4.—Map showing arrangement of drill-holes in a certain area. Solid circles show first drill les; open circles show second series of drill holes; dotted line shows area of pay gravel; dot-and-dash e shows deepest part of deposit.

Figures 4 and 5 represent prospecting maps of two different reas of dredging ground and show the method of arranging drill oles. These areas were purchased and equipped as a result of the rospecting work indicated by the maps. The illustrations show ctual and not theoretical field conditions.

Sometimes it is advisable to disregard the results for certain sections, perhaps one end of a property under examination, the results r the remainder indicating that its size and gold content warrant development, whereas, if the results for all the holes drilled were averaged, the figure so reached might not indicate development to be warranted. There are instances in California where dredging of part of a property would have been profitable, but the necessity of having to purchase a large unproductive acreage in order to procure the desirable part made the project an unattractive mining investment.

In estimating the value of the gold contained in an area to be dredged, it is the general custom to take a percentage, generally 75 to 80 per cent of the total value indicated by the prospect drilling as indicative of the amount of the gold content recoverable by dredging. The percentages mentioned are based on several tests in the Oroville and other California fields.

The following reference to the Pato property is taken from the report of the Oroville Dredging Co., dated September 30, 1911, to show the prevailing practice in discounting the indications furnished by prospecting:

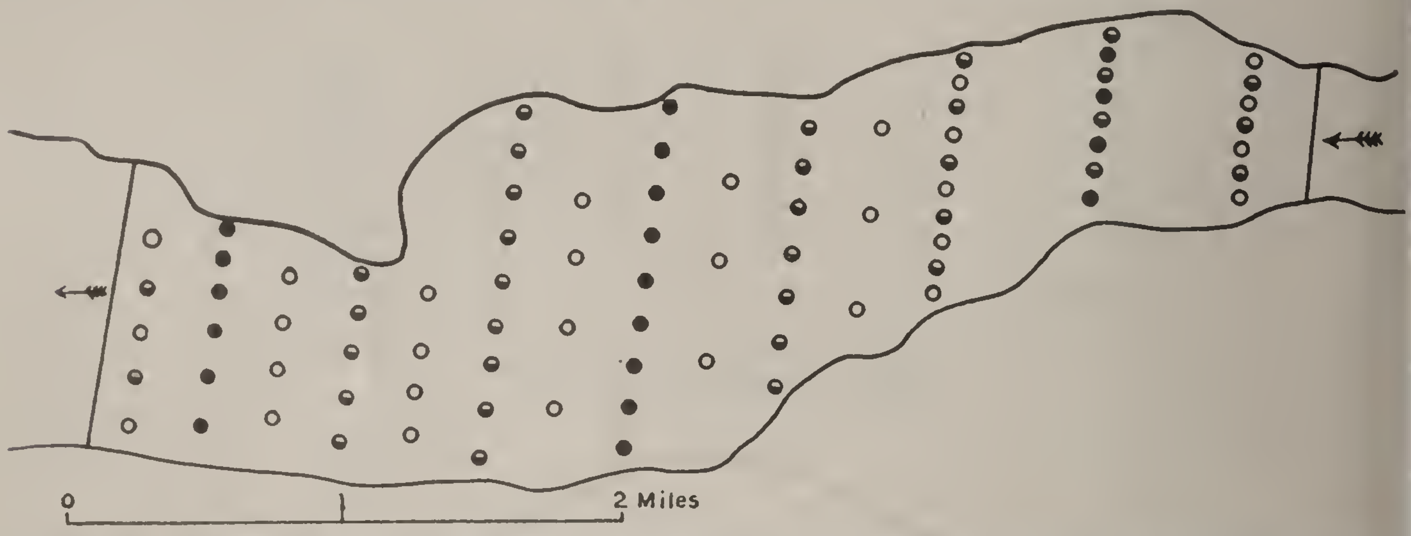
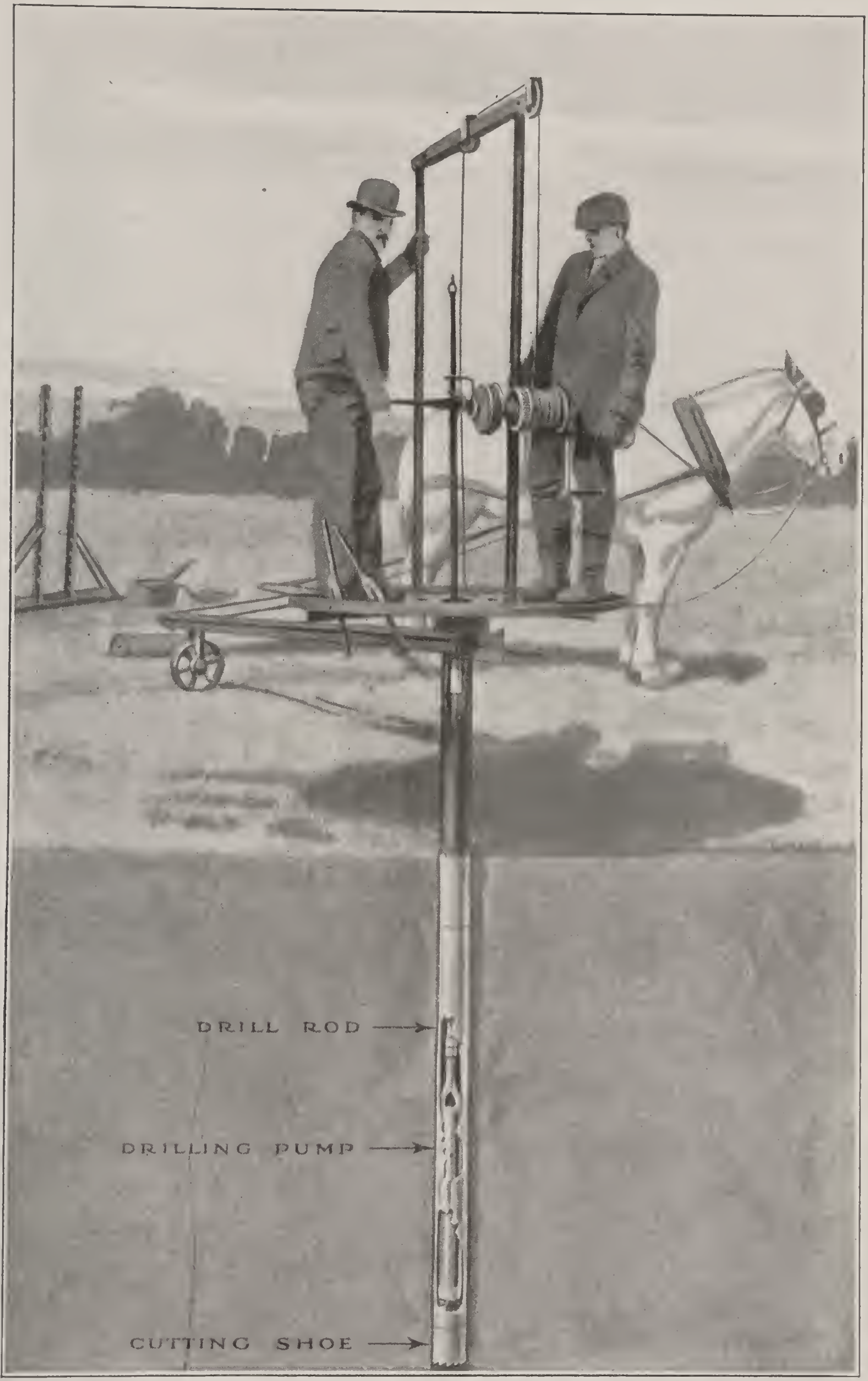


FIGURE 5.—Map showing arrangement of drill holes in another area. Solid circles show first series of drill holes; shaded circles show second series of drill holes; open circles show last series.

> The original estimates of the engineers give us an area containing 13,637,000 cubic yards of material yielding gross $3,202,986, there having been deducted from the values recovered by prospecting tests 25 per cent for variations, possible losses, etc.

It is, however, impossible to give any fixed percentage to offset the effects of various conditions of examination and operation.

The reliability of the tests and the accuracy of the average valu indicated either by drilling or by sinking shafts depend on the car taken in prospecting, the number of samples procured, the location o the holes in relation to the deposit, and the experience and abilit of the men in charge. If these factors are satisfactory, the estimat of the value of the property should require no discounting other tha to recognize that the recovery of the dredge is always less than th gold content indicated by prospecting and that the returns on th investment should be enough to compensate for any risks taken After the gold content of the property has been estimated it is necessary to determine whether the returns over working expenses an



WORKING PARTS OF EMPIRE PROSPECTING DRILL.

amortization of capital invested will allow sufficient dividends to make investment attractive.

TYPES OF HAND DRILLS.

The calculations outlined are based on sampling with a Keystone or similar power drill. However, in recent years the hand drill has been used extensively with great success, particularly in preliminary tests where the lower first cost of the hand drill and its lighter weight, which permit transport by either pack animals or men, are important considerations.

EMPIRE DRILL.

Of the hand drills in use in the United States and Alaska, the Empire drill (Pl. XI) is the most widely used. Briefly, the drill consists of a pipe, a platform, the rotating and driving devices, and the boring, drilling, pumping, and pulling tools. The operation is as follows:

The pipe, having a toothed cutting shoe on the bottom and a platform on the top, is placed in an erect position at the spot where the sample is to be taken. Where practicable, a small auger the size of the casing is generally used to start the hole. The material removed by the auger is washed with the rest. Some of the crew steady the pipe and others stand on the platform and with a battering ram drive the pipe into the ground until it will stand without help. The rotating device is then attached and a combination drilling and pumping tool screwed to the rods is used as a churn drill, the pipe being rotated simultaneously. While it is thus kept loose in the ground, it is sunk (1) by the weight of the pipe, (2) by the weight of the platform attached to the top of the pipe, (3) by the weight of the drill men who stand on the platform, (4) by the weight of the rod and tools that the drill men operate while standing on the platform, (5) by the jarring effect of the tool striking on the bottom of the hole. As the pipe sinks into the ground a core accumulates, and the churning crushes this core and forces it into the pump. Despite the contention of manufacturers, the hand drill usually requires more than rotation and weight to sink it and driving with a battering ram will almost always be necessary.

When the pump is filled it is removed from the pipe and its contents are emptied. It is again lowered into the pipe and the operation repeated. The contents are washed and the estimated weight and the characteristics of the gold are noted. These operations are repeated, lengths of pipe and drill rod being added as required, until bedrock is reached. The pipe is then pulled from the hole by leverage, and the apparatus is moved to another spot, and another hole is started. The drill acts like a combined core drill and churn drill, for the toothed shoe cuts a core and the drilling and pumping tool performs the work of a churn drill as well as of a pump.

Calculations are based on the same plan as for the power drill, except that the diameter of the pipe of the Empire drill is smaller than that of the power drill. The material handled with the power drill is nearly two and one-half times the amount handled with the Empire drill; that is, the dirt recovered during 238 feet of drilling with an Empire drill would represent 1 cubic yard, whereas the dirt from 100 feet of drilling with a Keystone drill would approximate this amount.

Under favorable conditions the speed of the Empire drill exceeds that of the Keystone, and its ease of transportation is at times an important feature (see Pl. XII). Where labor is cheap, especially in some foreign countries, the cost of drilling with a hand machine is much less than that with a power drill.

Hand drills of the percussion type proved expensive and impracticable when used in Alaska. In examining shallow deposits, 6 to 10 feet deep, it is at times possible to improvise, at small expense, a hand-drill equipment that will give a rough indication of the gold content of the ground. Thus, a piece of casing may be driven directly to bedrock, then pulled, and the core may be washed. Needless to say, the sample obtained has not the value of a sample obtained with either of the two drills mentioned above.

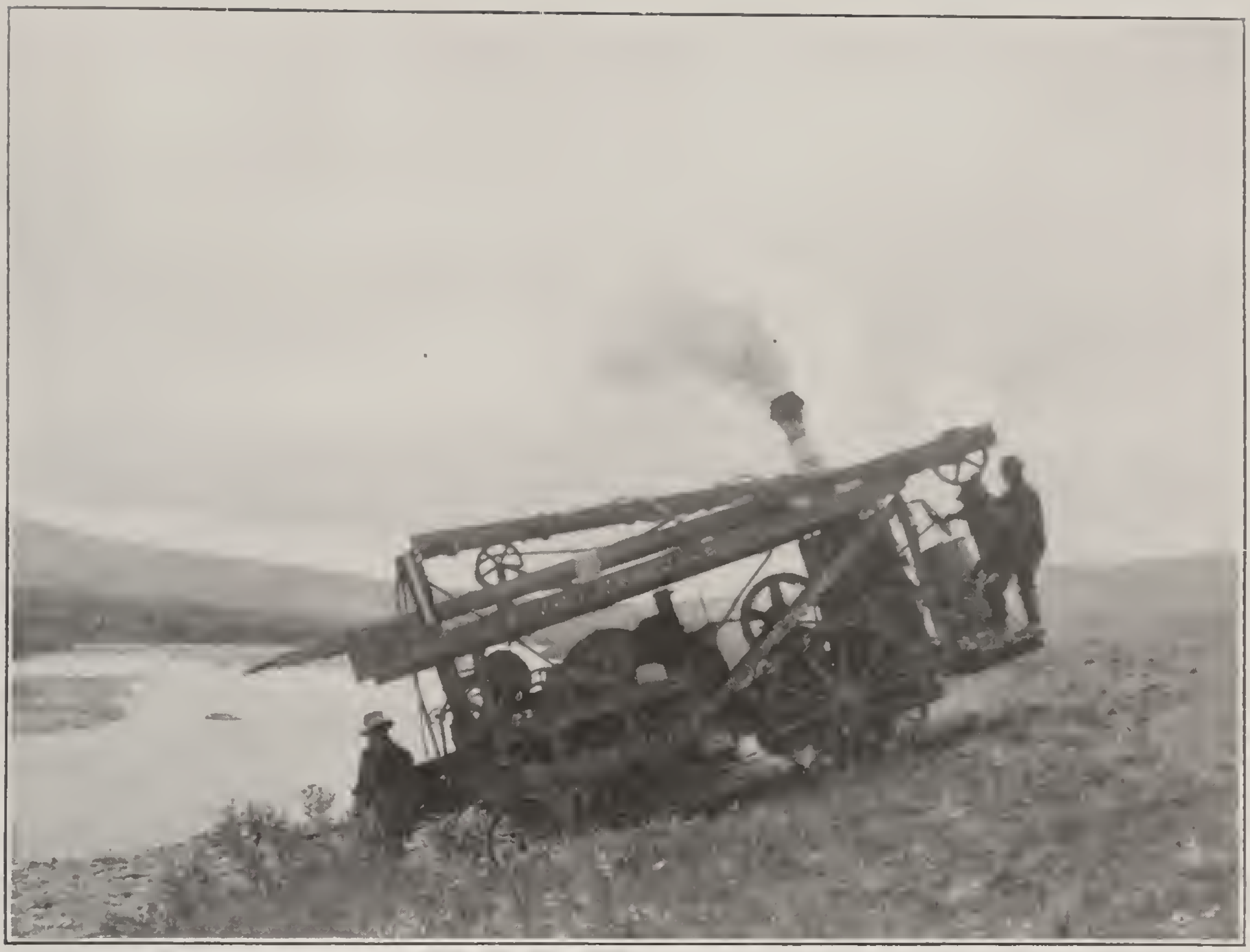
NEW TYPE OF LIGHT POWER DRILL.

A type of light power drill (Pl. XIII) has recently been put on the market and is favorably regarded by some operators and engineers. It is a drill of the percussion type, somewhat similar to but smaller than the Keystone drill, and is operated by a small gasoline engine. It is light, and when erected on a property is easily moved to a new location.

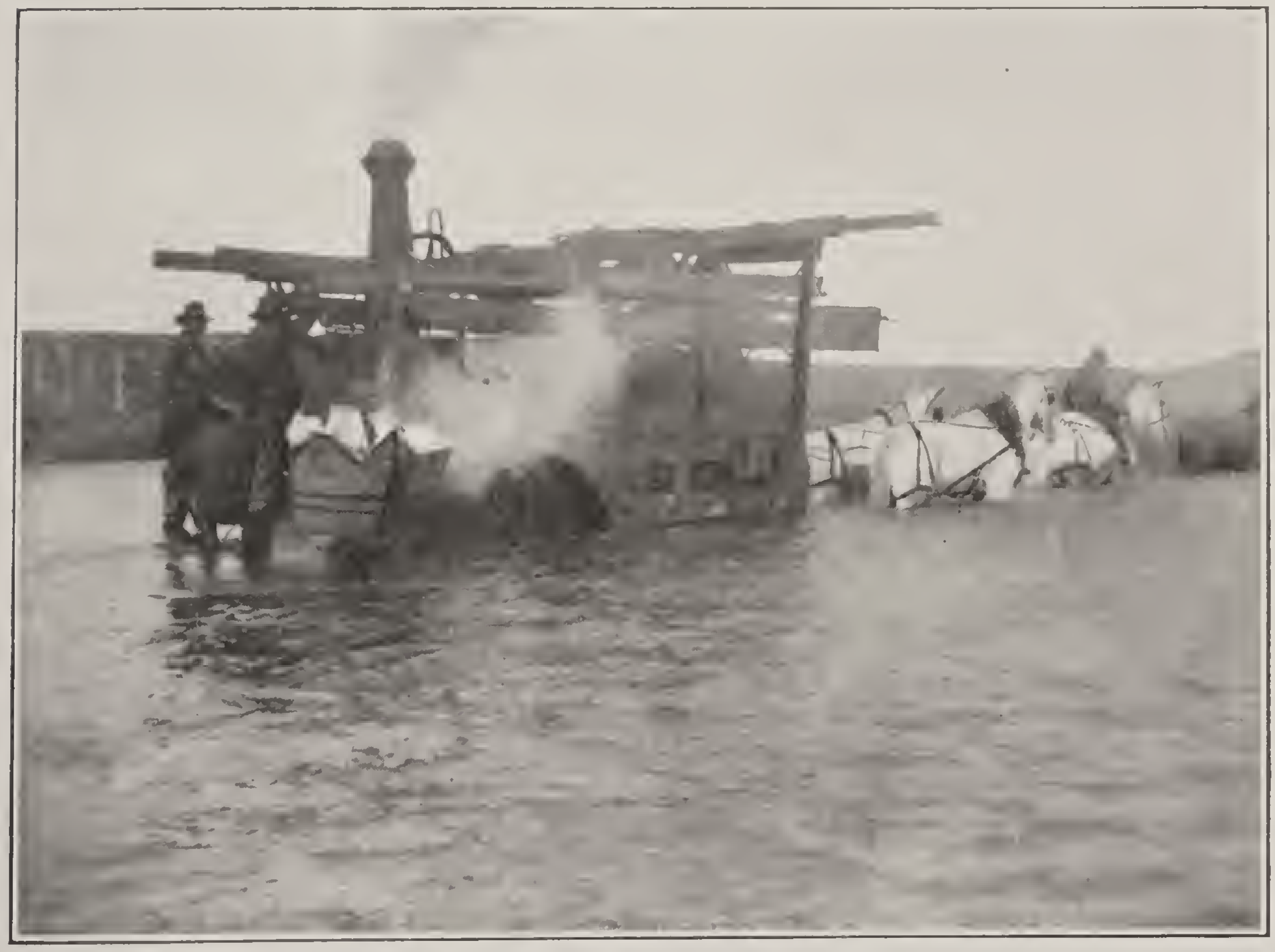
DETERMINING PERIOD OF DREDGING PROJECT.

The life of a dredging property can be estimated much more closely than can that of a vein mine. Also, the ore in sight in vein mining is figured as a guaranty for the return of only a part of the capital invested, whereas in gold dredging allowance must be made for recovery of the cost of the property and the equipment during the life in sight. The dredge equipment is usually of sufficient capacity to turn over the ground in 7 to 10 years, as that period is generally conceded to represent the life of a dredge with a wooden hull, although a few such hulls have lasted more than 13 years and are still in use.

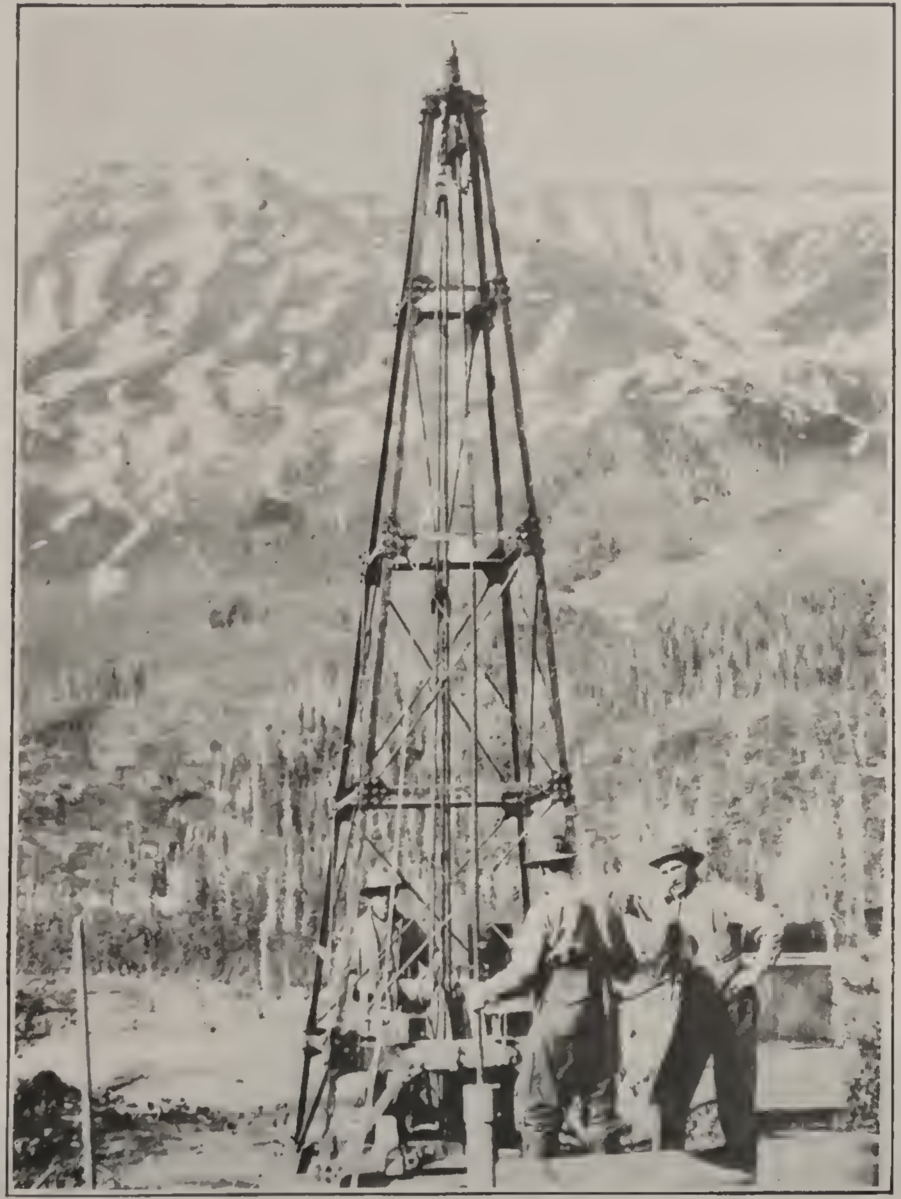
Although the rate of interest desired on the investment varies with the ideas of different operators, it is generally conceded that after allowing for amortization of capital the minimum profit to be figured for a dredging project is 10 per cent per annum.



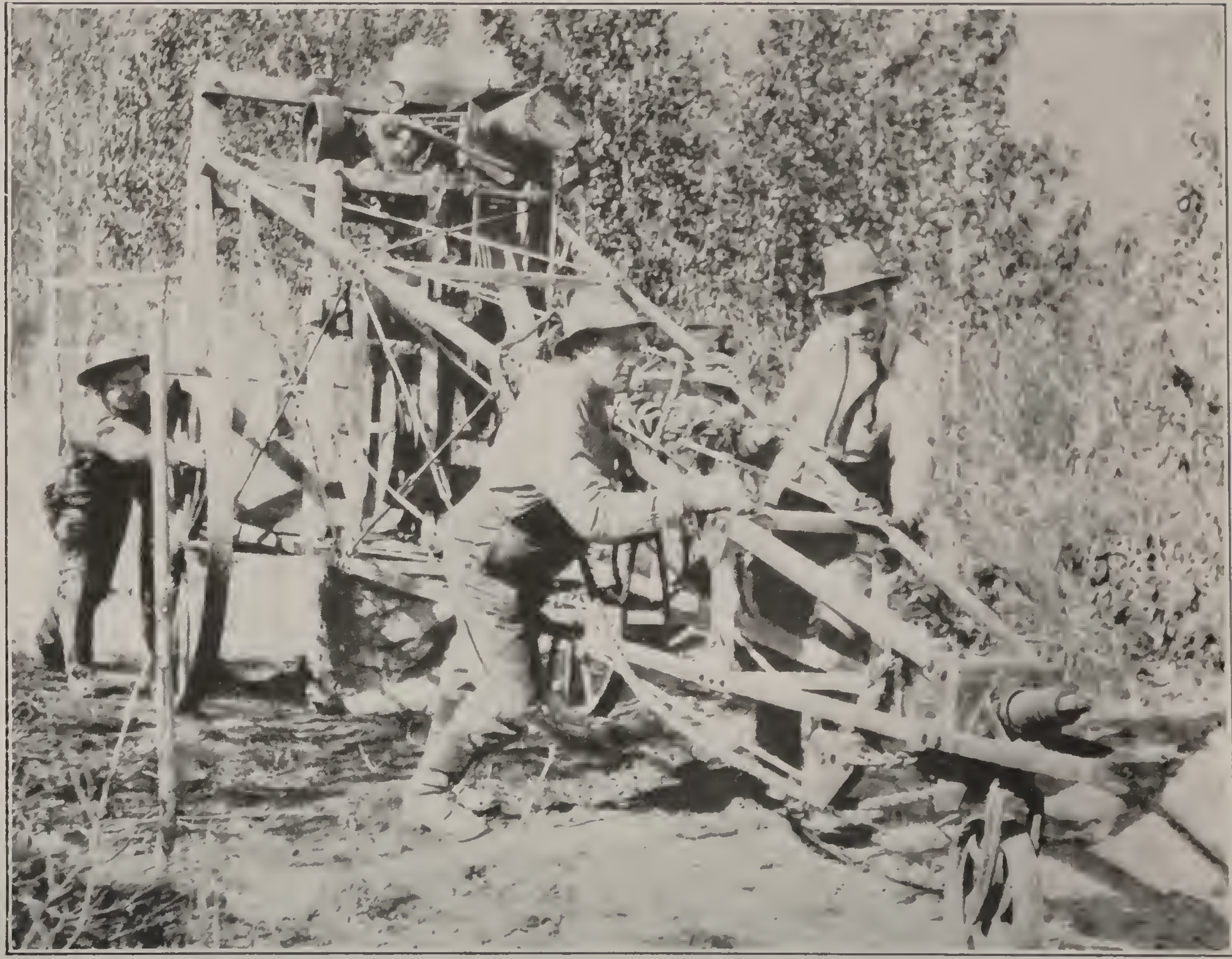
A. KEYSTONE TRACTION DRILL DESCENDING A RIVER BANK.



B. KEYSTONE NONTRACTION DRILL FORDING A RIVER.



A. OPERATING LIGHT POWER PROSPECTING DRILL.



B. RECENT TYPE OF LIGHT POWER DRILL BEING MOVED.

From the working results of different size dredges over a considerable period, as shown in the section on working costs, the time required to work a given area of ground by different size dredges and the working costs under average conditions in California can be closely approximated. The following example will indicate how these figures can be applied.

An acre contains 4,840 square yards, so that an acre 11 yards deep would contain 53,240 cubic yards, which, with allowance for ground lost in turning and for uneven bedrock, can be figured at 50,000 cubic yards. If an examination of an area on which dredging is projected indicates 500 acres of workable ground 11 yards deep, the contents in cubic yards would be estimated as 25,000,000.

Under favorable conditions a boat having a capacity of 15 cubic feet would work out about 500 acres 11 yards deep, or 25,000,000 cubic yards of gravel of moderate compactness, in about 8 years, allowing 80 per cent working time, at a total operating cost of 4 to 5 cents per cubic yard. To work out this same area, two 7-cubic-foot boats would require 10 years, at a cost of 5 to 7 cents per cubic yard, and three 5-cubic-foot boats would require 11 years, at a cost of 5½ to 8½ cents per cubic yard. The cost given per cubic yard includes the total working and administration charges.

SELECTION OF DREDGE FOR A GIVEN PROJECT.

Decision as to the most desirable size of boat for a projected dredging operation sometimes requires considerable study, and here again is where the judgment of the experienced engineer is valuable. It is safe to say that the experience of the California operators is such that under certain conditions the large 15-foot boats would not be recommended as the most economical. Where compact and cemented gravel predominate, probably a 9-foot or 10-foot boat would now be selected for such work, partly on account of lower first cost, but more especially on account of expenses and loss of time for repairs with the large boats, and the difficulty of handling the heavy parts.

No hard and fast rule can be prescribed for determining the best size of dredge, and because of the development of the gold dredge to its present efficiency, the question is often raised as to when the maximum size for economic installation will be reached. All depends upon operating conditions. Under the favorable conditions at Idaho City, the 18-foot bucket of the No. 4 dredge of the Boston & Idaho Co. demonstrates the economic efficiency of the large dredge. There is no question as to the mechanical possibility of buckets even larger than those now in use, as is discussed under "Reclamation of dredged land," page 205.

In addition to the gold to be recovered from the gravel, the probable value of land reclaimed for agriculture might, in some regions, be considered in estimating the total profit.

Any estimate of the working cost of a dredge of this character is, of course, pure speculation, but there seems every reason to expect that, under favorable conditions, or in redredging some previously dredged areas, a very low operating cost would be obtained.

USE OF UNUSUAL TYPE OF DRILL IN ALASKA.

Contract drilling for a placer examination is not usual, nor under most circumstances is it wise. The following description of an unusual kind of drilling machine employed on contract during an examination on the Seward peninsula in 1912 is of interest. A steam drill on the property was found unsuitable, partly because of the soft nature of the tundra over which it had to be hauled, and partly because the machine was an inferior type and in poor repair. Hand drills of the percussion type were tried but were not successful. The cost of drilling, on account of frozen ground and of high wages ($7 a man daily) was excessive, as only a few feet a day could be drilled. The drill procured for the work was made by a contractor who had had considerable experience with drilling in frozen ground. It was in type and action similar to a Keystone drill. Power was furnished by a gasoline engine of about 25 horsepower. The width of tread of the machine was 16 feet 6 inches; the drive wheels were 6 feet wide, being designed for the soft tundra. The drill could be driven under its own power over the tundra where in places even walking was difficult and also over tailing piles and uneven ground. The drill stem was made of two ordinary Keystone stems, and a heavy 6½-inch bit was used; the weight of bit and stem was estimated at 1,850 pounds. In order to obtain the services of the contractor, it was necessary to enter into an agreement for the drilling. Under different circumstances, considerable risk might be assumed in drilling placer ground by contract, but care was taken to see that the terms of the contract were such that the operation was entirely under the direction of the engineer, and moreover the contractor's references for ability and reliability were excellent. A minimum of linear drilling distance of 2,500 feet was guaranteed, with the right to drill up to 5,000 feet additional if desired.

The contractor furnished the drilling machine and all supplies for it and agreed to take personal charge of the drilling and to furnish helpers. He also supplied board for his men and made repairs. The engineer furnished panners and rocker men (two to each shift). In addition either the engineer or an assistant was in charge of the work during the entire time. The terms of contract were 80 cents a foot for frozen ground and $1.50 for thawed ground. Casing was necessary in thawed ground, but not in frozen ground except for one length of pipe at the commencement of each hole. There was approximately 2,500 feet of drilling done from June 28 to July 29. One shift was employed at first until the drill, which had been out of commission for some time, had been put in good running order. The second shift commenced about July 3. The shifts were of 12 hours each. Wages paid for panners were 50 cents an hour and board.

The rate of drilling, averaging a little more than 40 feet a shift, was far less than the contractor usually obtained. He stated that he generally averaged 100 feet or more a shift. The lowered rate was caused by the greater care insisted upon in drilling, by the time required for pumping every 2 feet in the upper part of drill holes, and every foot when colors were encountered as compared with every 5 feet in all ground, the usual requirement there. A greater number of pumpings for each section were also taken than was usual at Nome. To test the reliability of the pump, several customary tests were made, such as pouring a certain number of small shot in the hole, and obtaining the same number after pumping.

The action of the drill was similar to that of the Keystone. Pumpings were dumped into a rectangular zinc box and from there washed with fresh water to pans held over galvanized tubs; all dirt recovered from each section was panned separately and the number of colors counted; afterwards all tailing from the panning was rocked, and any gold recovered from rocking was added to that from panning. The rocker tailings were frequently sampled to ascertain whether there was any loss of gold. The volume of dirt from each hole was measured by digging a hole 3 feet square and 1½ feet deep, at the end of the rocker. The tailing from the rocker was run into this hole and allowed to settle and the cubic contents afterwards determined. From 3 to 5 pumpings were made from each section. Sections were taken every foot in "lower" ground, and every 2 feet in "upper" ground or overburden. The bit was measured before and after each hole was drilled and an average was taken in calculating the volume taken out. The figure so obtained was checked by measuring the dirt from the hole, and a proper factor for calculations was ascertained. It was impossible to get the best results with one panner; hence two were used on each shift and were generally kept busy.

It was not easy to arrive at the total cost of sampling on account of difficulty in properly proportioning the time of the men. Actual charges against the drill, engineering fees, etc., being omitted, amounted to about $1.20 a foot.

DETAILS OF CONSTRUCTION OF GOLD DREDGES.

DETERMINING SUITABLE SIZE OF DREDGE.

After the ground has been prospected, the selection of a dredge best suited to the work is the next important step, provided the results of the examination have been favorable. Some costly errors have been made; in fact, in some projects the selection of the wrong type or size of dredge has meant failure instead of success. In general it may be said that if the ground is deep, the gravel not cemented, and the property large, then the larger the dredge the lower will be the cost of operation. On the other hand, if the ground is shallow, say, 30 feet or less, a large dredge is undesirable, as the hull for a large dredge would have to be 10 or 12 feet deep and its flotation would make trouble. For the same reason, if the ground is shallow, the grade of the bedrock is also a factor in determining the size of the dredge. If the ground is tightly cemented the buckets should not be of the largest size but should be small and strongly built. Where bowlders are present the large buckets make for ease of handling.

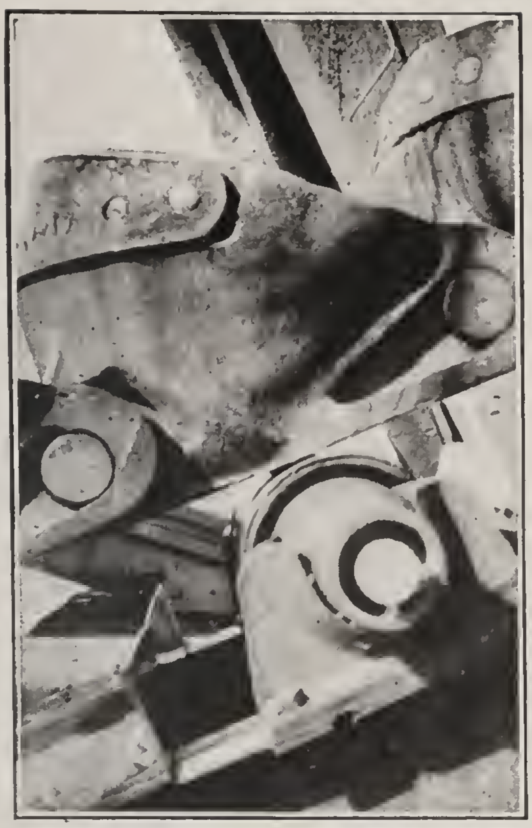
When the ground can be dug without undue difficulties, the production with different sizes of buckets should be about as follows:

Probable production with various sizes of buckets in fairly loose ground.

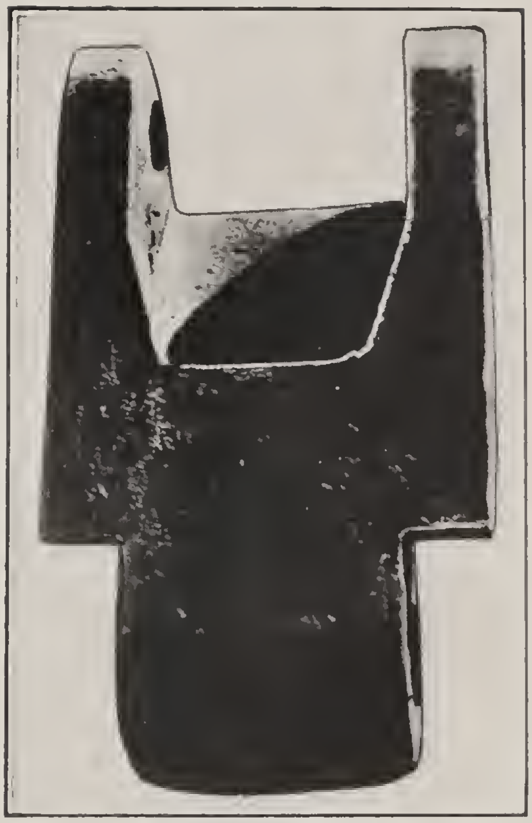
Capacity of buckets, cubic feet.	Monthly production, cubic yards.
3	50, 000
5	85, 000
7	120, 000
9	180, 000
16	300, 000

BUCKET LINE AND TUMBLERS.

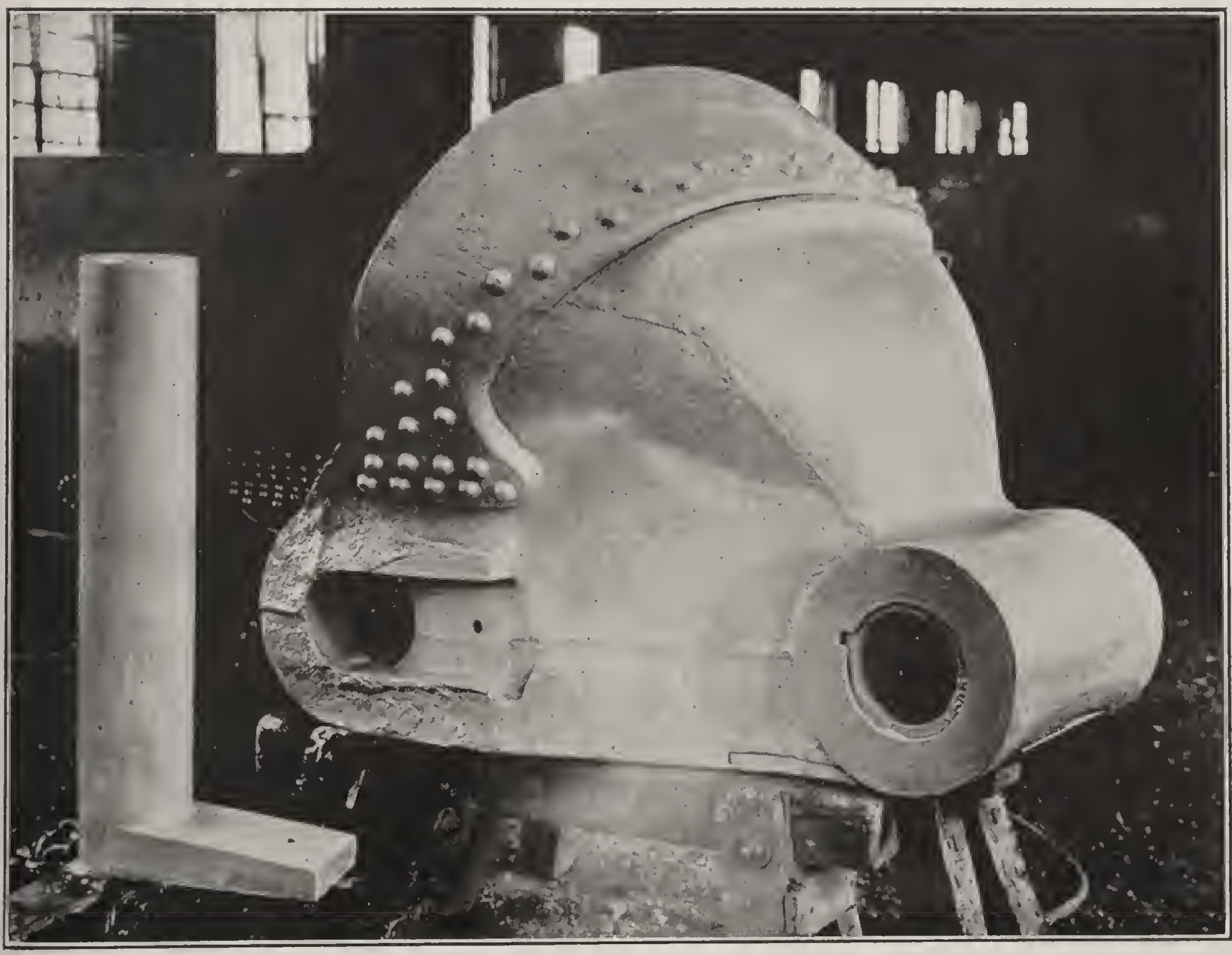
The bucket line and tumblers still present the greatest difficulties to the designer and operator, and the fact that on practically every new dredge experiments are made with new types is evidence that there is still a wide difference of opinion as to the best design. It is conservative to say that the bucket lines of more than half of the dredges going into new fields are failures, or at least entirely unsuited to their work. And when it is realized that the bucket line of some large dredges costs $50,000 or more installed, the seriousness of such failures becomes evident. Many failures can be traced to the design of the equipment which has not been suited to the ground to be worked.



A. LOW-BACK BUCKET FOR DIGGING BOWLDERS; ALSO SHOWING LADDER ROLLER.



B. BASE OF NEW TYPE BUCKET FOR USE WITH ROUND LOWER TUMBLER.



C. MANGANESE STEEL BUCKET OF 16 CUBIC FEET CAPACITY, CONREY NO. 4 DREDGE; SHOWS FULL CIRCULAR BUSHING IN BACK EYE.

The character and the depth of the ground to be dug as well as the size of the gravel determine the shape and weight of the bucket. A bucket well designed for working in small gravel would be unsuited for bowlders and vice versa. When much clay is present still another shape would be more suitable. A bucket to handle bowlders (Pl. XIV, *A*) should be designed with a low back so that when bowlders are picked up and the bucket carrying the rock comes around the lower tumbler to the horizontal position the bowlder will not crush the back of the preceding bucket. Plate XV shows a dimensioned drawing of a 16-foot bucket.

The buckets should not be filled so full that excessive spilling will occur. Some ground is of such a character that the bank stands firm and it is necessary to dig to the top; in such digging the bucket line is almost in a horizontal position, and all gravel in excess of the natural repose of the material will wash out, part of it being lost over the side of the ladder, and the remainder being returned by the ladder pan to the front of the ladder and redug, resulting in a loss of efficiency. Hence the lip of the bucket should not be much higher than determined by the natural angle of repose of material.

In many fields the top of the bank caves, so that the ladder of the dredge is always kept at a considerable depth. In such digging the lip of the bucket can be higher than when the bucket line is horizontal. The bucket should be as high as the factors mentioned above will permit, as of course the higher the bucket the greater the capacity.

The present practice is for the buckets to have two front eyes and one rear eye (Pl. XVI, *A*). The so-called "three-eye buckets" with three eyes in front and two behind have been discontinued, as the lugs broke readily and the pins were taken out with difficulty. They were more expensive to manufacture and it was necessary to design the inside of the buckets with pockets which interfered with the dumping of the material. The idea underlying the design of these buckets was that owing to the three supports the pins would not break so readily.

The width of the bottom of the bucket should vary directly with the length of the bucket line to be used, and also with the hardness of the material to be dug (Pl. XIV, *B*). It is also necessary to have sufficient bearing area so there will not be excessive wear where the bucket seats on the upper tumbler. In order to increase this bearing surface the "square bottom" bucket is now being largely used. This type has several other advantages. The wear on the bucket where it bears on the lower tumbler is reduced, as is also the wear on the ladder rollers. The wear on the flanges of the lower tumbler and the side plates on the upper tumbler is negligible, whereas with the old buckets considerable trouble was experienced at these points. Plate XVI, *B*, shows the eye of a bucket elongated by wear.

If the material to be dredged is clayey the bucket should be wide and short to facilitate dumping. The buckets on the early dredges were long and narrow and when any sticky material was encountered it would not be discharged. This trouble was increased on headline dredges in early days, as the clay was dug in a mass. The present practice in handling clay is to swing the dredge by means of spuds and to dig lightly so that the material enters the bucket in comparatively thin slices.

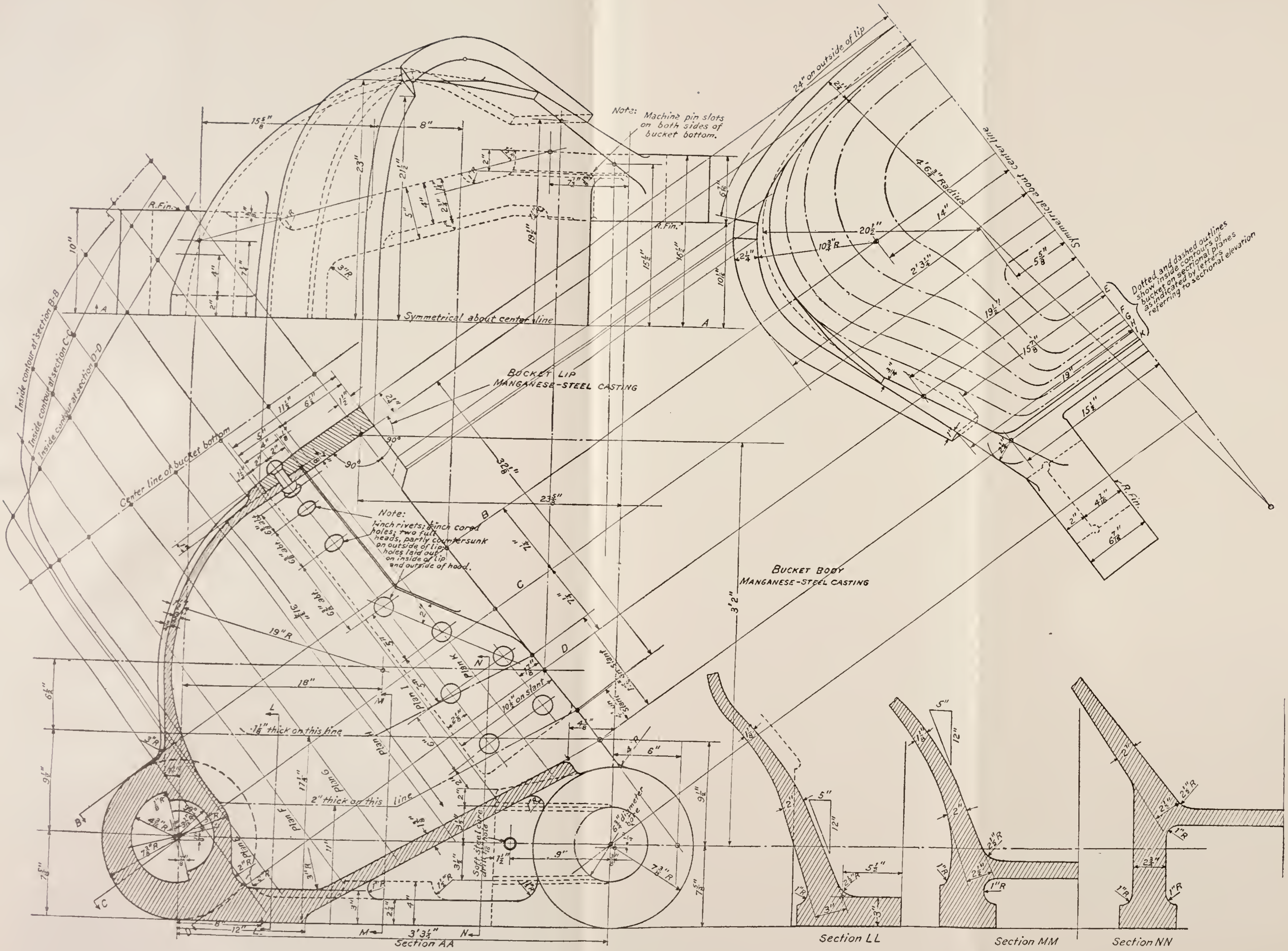
The original California type of bucket consisted of three parts—the bottom, the hood, and the lip (Pl. XVII). The bottom was always cast, either of special carbon, manganese, or nickel-chrome steel. The hood was pressed or cast steel, and the lip was manganese steel. This design necessitated a large number of rivets which frequently became loose and caused trouble. During the past 10 years the art of making steel castings has been so developed that it is possible to cast the hood and bottom in one piece (Pl. XVIII, *A*), which does away with leakage and is cheaper in first cost. With this type it is more difficult to cast the lips to make an accurate fit on the cast hood, but experience is remedying this drawback. It is important to have the lip fit snugly to the hood, so that the lip will not have to be drawn in with bolts before being riveted, as the spring of the lip will put an initial strain on the rivets that will ultimately cause them to become loose. It is generally better to have a few large rivets than a greater number of small ones. On 16-cubic-foot buckets 1¼-inch rivets are not too large and 7-foot buckets should have 1⅛-inch rivets. Tendency of the rivets to shear should be obviated by having the lip butt against a lug on the bucket.

A common error in bucket design is making the arch of the lip too flat. This fault frequently leads to crushing of the lip.

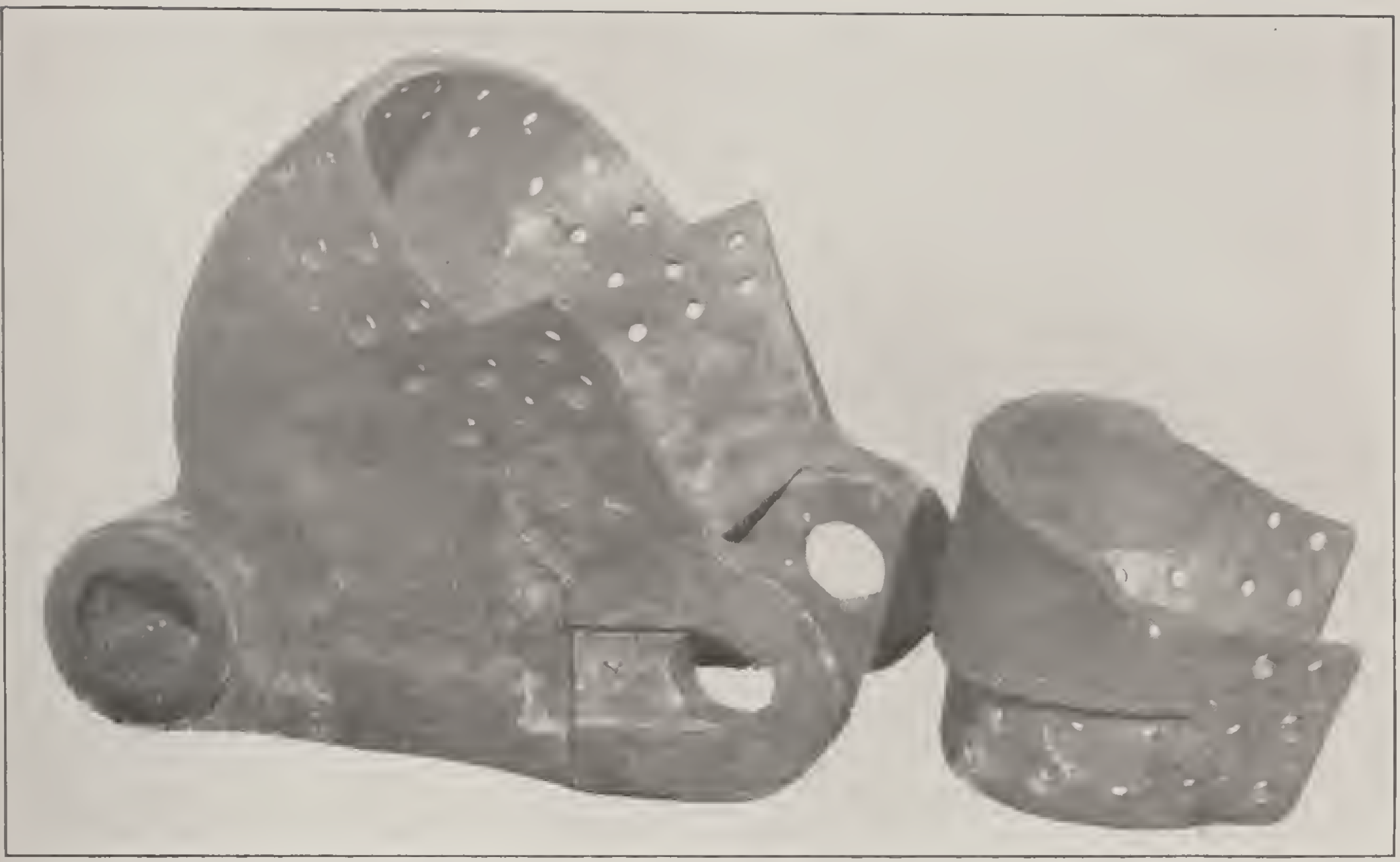
When the hood and the bottom are cast together they are usually made of manganese or nickel-chrome steel. When made of nickel-chrome steel a manganese-steel insert plate (Pl. XVI, *C*), is peened into the bottom of the bucket where the wear comes on the upper tumbler. Care should be taken that this insert is set ahead far enough so that the corner in the recess that takes the insert plate will not come near the corner of the bushing seat as a crack is likely to occur between these points. A large proportion of buckets that fail break at the back eye whether they have insert plates or not, but the plate is an additional source of trouble. Plate XVIII, *B*, shows a 2-cubic yard bucket used on a Panama Canal dredge, and Plate XIX, *A*, shows a 6-foot bucket used on the Guamos dredge in the Philippine Islands.

PINS.

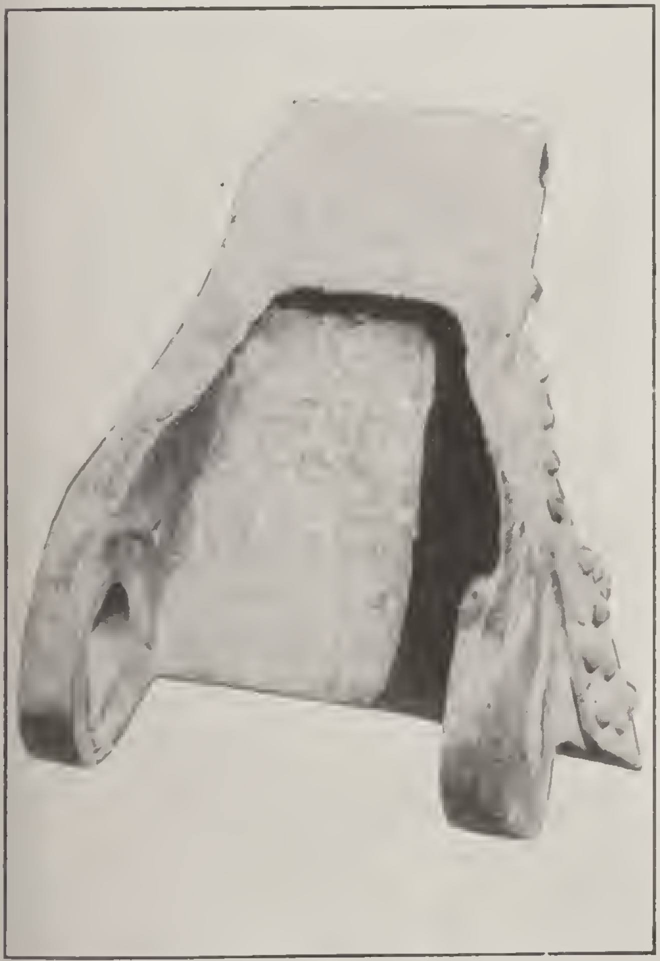
The diameter of the bucket pins should not be limited to the size necessary to withstand calculated breaking strains, for wear of the pins has been found to be the determining factor. It is true that


Note: Machine pin slots on both sides of bucket bottom.
24" on outside of lip
Symmetrical about center line
4'6¾" Radius
20½"
10¾"R
14"
2'3¼"
5⅝"
19½"
15⅞"
19"
15¼"
Dotted and dashed outlines show inside contours of bucket on sectional planes as indicated by letters referring to sectional elevation
E
F
G
H
K
R.Fin.
2"
4 7/16"
6 7/16"
15⅝"
8"
23"
21½"
R.Fin.
10"
Symmetrical about center line
A
Inside contour at section B-B
Inside contour at section C-C
Inside contour at section D-D
Center line of bucket bottom
Bucket Lip
Manganese-Steel Casting
90°
32⅛"
23⅝"
B
C
D
Note: ⅞-inch rivets; 1-inch cored holes; two full heads, partly countersunk on outside of lip; holes laid out on inside of lip and outside of hood.
Bucket Body
Manganese-Steel Casting
3'2"
19"R
18"
10½" on slant
1⅛" thick on this line
2" thick on this line
L
M
N
plain
Soft steel core drift 1/16" hole
6¾" diameter bore
7⅜"R
9"
6"
3'3¼"
12"
Section AA
Section LL
Section MM
Section NN
5"
12"
5½"
1"R
2⅝"R
2"
1½"R
2¾"

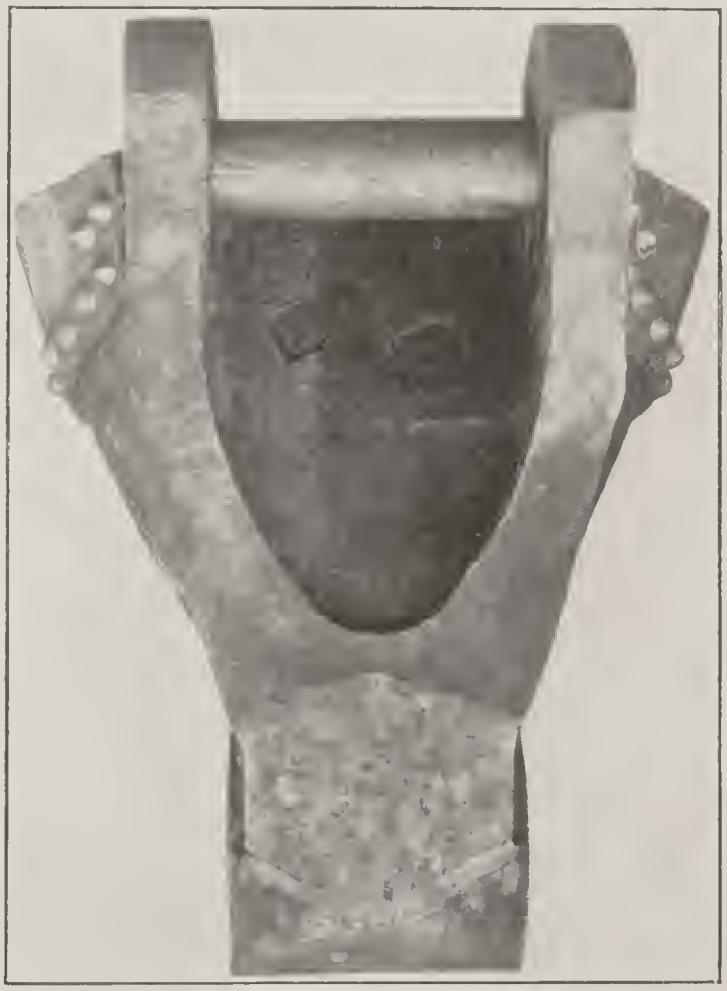
DETAIL DRAWING OF A 16-FOOT BUCKET.



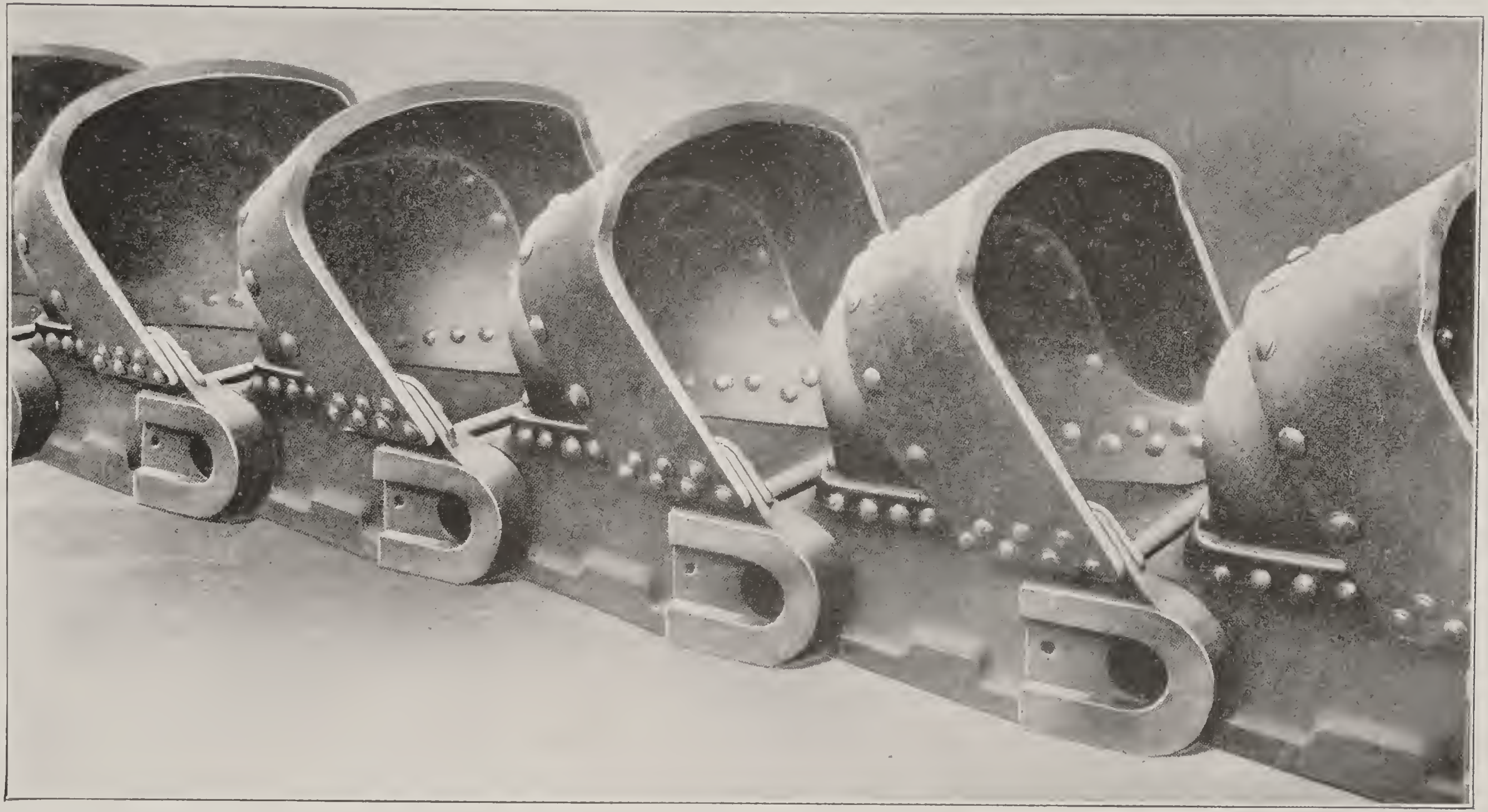
A. ONE-PIECE MANGANESE-STEEL BUCKET WITH SPECIAL TYPE OF LIP.



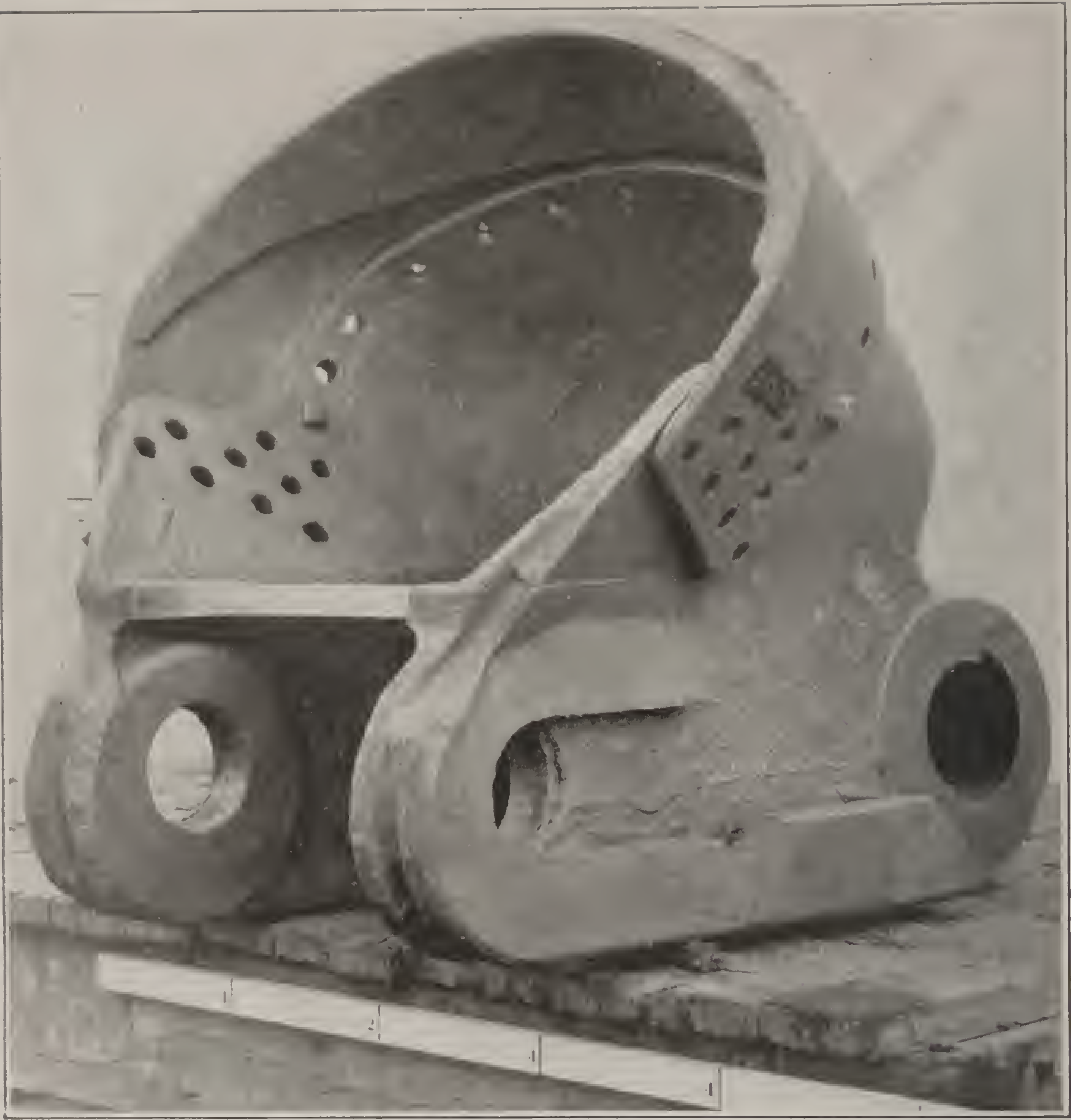
B. BUCKET WITH EYES ELONGATED BY WEAR.



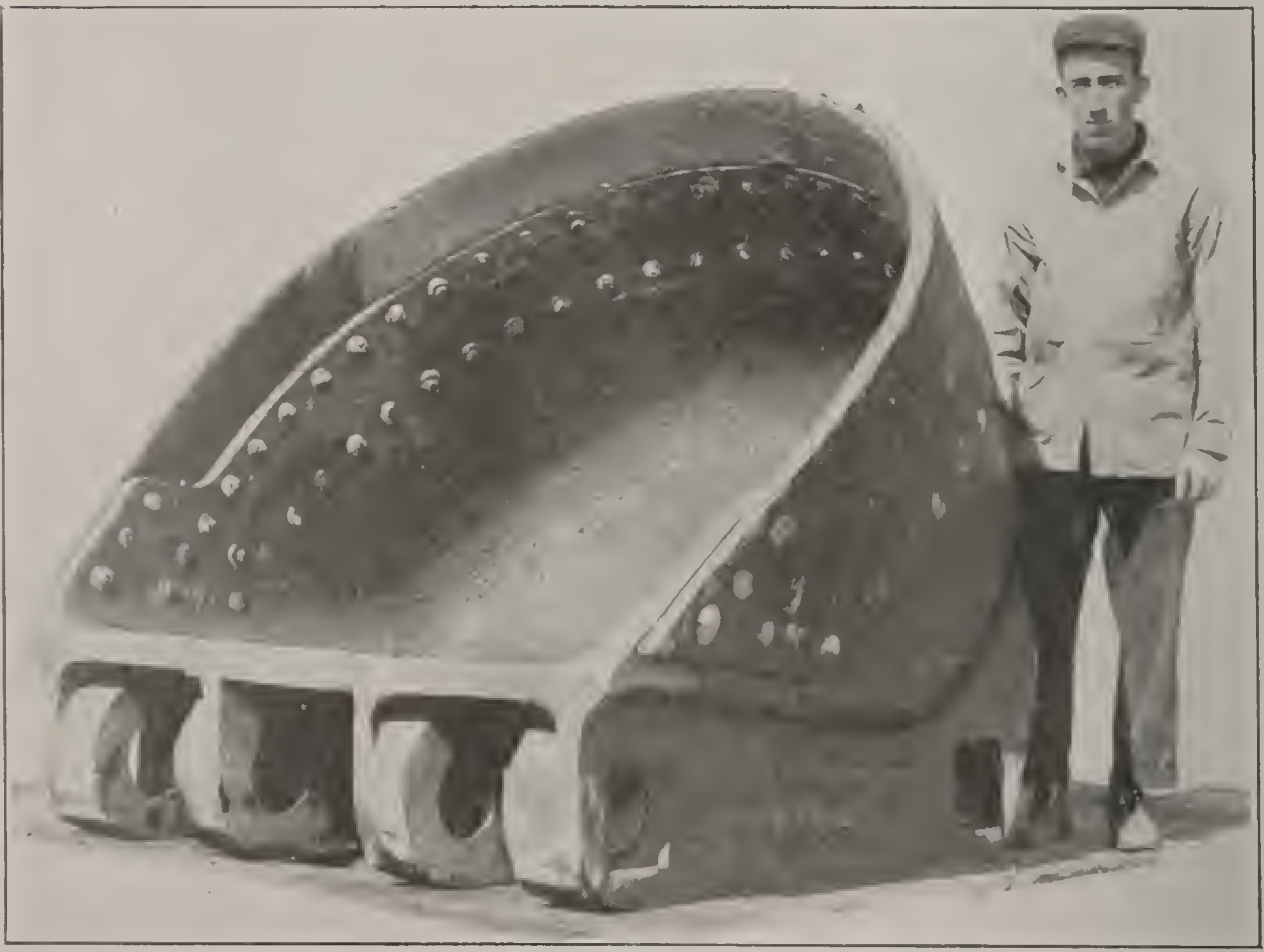
C. BUCKET WITH INSERTED PLATE OF MANGANESE STEEL TO RESIST WEAR ON TUMBLER.



ORIGINAL CALIFORNIA TYPE OF BUCKETS.



A. RECENT TYPE OF 16-FOOT BUCKET WITH BOTTOM AND HOOD CAST AS ONE PIECE.



B. BUCKET USED ON A PANAMA CANAL DREDGE; CAPACITY, 54 CUBIC FEET.

pins often break in service, but most breakage is due to inefficient heat treatment and not to small diameter. The chief cause of bucket-line trouble is the elongation of the pitch of the buckets and the decrease of the effective pitch of the tumblers owing to wear. Not only does a pin wear but it wears the bucket eyes and the bushings. Hence the larger the pin the longer will be the life of the bucket. In the standard type of bucket the front eyes are not designed with the idea of having the pins turn in them but the pins are held against turning by lugs. A bushing is placed in the back eye to take the wear when the bucket turns on the pin. The half bushing is in most general use but some circular bushings are employed (see Pl. XIV, *C*, p. 54).

The lugs on the pins should be long and thick and fit snugly in the lug seats; otherwise the play resulting from looseness will soon cause the front eye of the bucket to wear rapidly. Pins are usually made of forged and tempered nickel-chrome steel, although manganese pins, machine ground, are being used with success. Several methods have been used to hold the pin in position but lugs only have proved successful.

Another type of bucket used on two or three dredges in California is designed so that the two front eyes have a combined length equal to the back eye. The pin is a straight piece of shafting free to turn in either the back or the front eye. Both the front eyes and the back eyes of this bucket have circular bushings.

LOWER TUMBLER.

The round lower tumbler (Pls. XIX, *B*, and XX, *A*) is the greatest improvement made in gold dredges in recent years, and as it is or will be universally used none of the other various types is here discussed. Most of the round tumblers at present are of manganese steel made in one piece. Some manufacturers make them of chrome-nickel steel owing to difficulty of producing a sound casting of this size and shape in manganese. Other companies make the tumbler in halves of chrome-nickel steel and use manganese wearing plates. The pressure used in setting in the shafts is about 125 to 175 tons, and it is not necessary to key the shaft in a round tumbler.

The following advantages apply to the round tumbler. With the hexagonal type (Pl. XIX, *C*), the difference in pitch of the bucket and tumbler is noticeable a short time after the bucket line is started and continues to increase until there is a difference of several inches. As the buckets ride around the lower tumbler, the slack caused by the difference in pitch accumulates for three or four buckets until the tension in the bucket line becomes great enough to cause the buckets to slide on the tumbler, and this slipping causes the wear on the tread and flanges of the tumbler. With a round tumbler such slipping is entirely eliminated. When the first experiments were made with the round tumbler it was expected to prove harder on the buckets and to cause them to break through the middle, as the buckets used in the experiments were worn and not designed especially for a round tumbler. It was also thought that a rounded surface would be worn on the bottom of the bucket where it had its seat on the tumbler, but such wear did not occur.

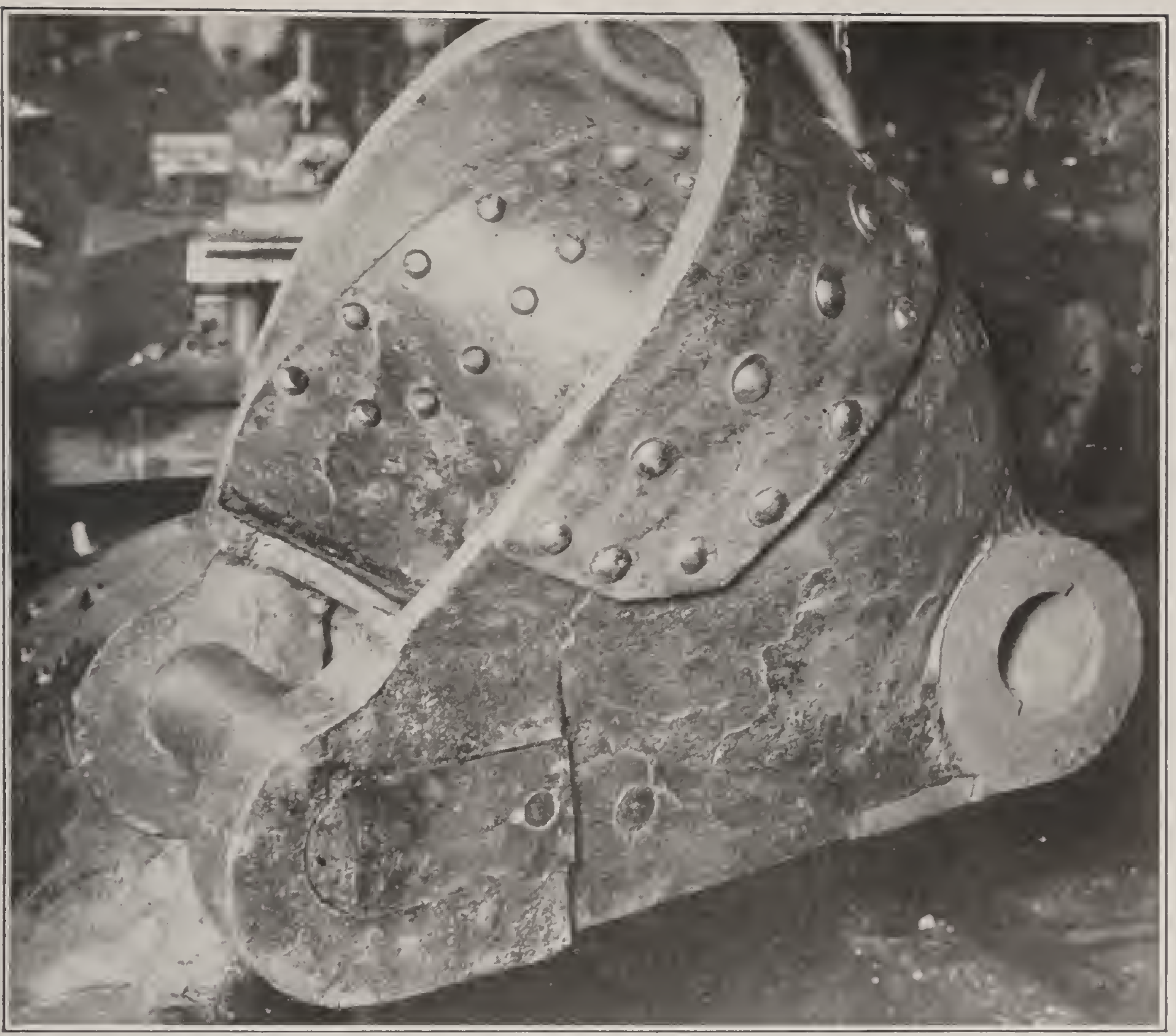
Round tumblers should be of such diameter as to obviate undue wear by the buckets. It is generally the custom to make the pitch of the tumbler a little greater than six times the pitch of the bucket. Thus, each successive bucket seats at a place different from that at which any preceding bucket has seated, causing the wear to be equally distributed all around the tumbler. Clearly, the efficiency of the round tumbler is not impaired by the elongation of the bucket pitch.

Tumbler shafts should be made enough larger than the bore of the tumblers to insure a tight fit. The oversize should vary from 1/1000 of an inch for each inch of diameter for shafts up to 6 inches in diameter to 1/2000 per inch of diameter for shafts 15 inches in diameter.

The shaft should be bored so that rods can be placed to hold the ladder from spreading. One method is to forge thrust collars on the end of the rod. If the ladder should spread when these are taken out the rods can be heated and put in place hot and the shrinkage will bring the ladder to the original position. Sometimes the tumbler shaft is turned so that collars are formed on the ends, and several other methods are used to keep the ladder from spreading (see Pl. XX, *A*, Pl. XX, *B*, and Pl. XXI).

There is still a difference of opinion regarding the most desirable type of lower tumbler bearing. Cast-iron bushings are generally used, but some companies prefer babbitt. Babbitt bearings do not wear so long, but they do not break as the cast bushings sometimes do, causing much damage to the shaft. However, opinion seems general that cast iron is preferable as the shaft wears more evenly. When a new bushing is to be installed the shaft is measured and the new bushing bored to suit the worn shaft. It is good practice to shrink nickel-chrome wearing sleeves on the shaft, as they wear at about the same rate as the bushings, and the sleeves and bushings can be renewed together.

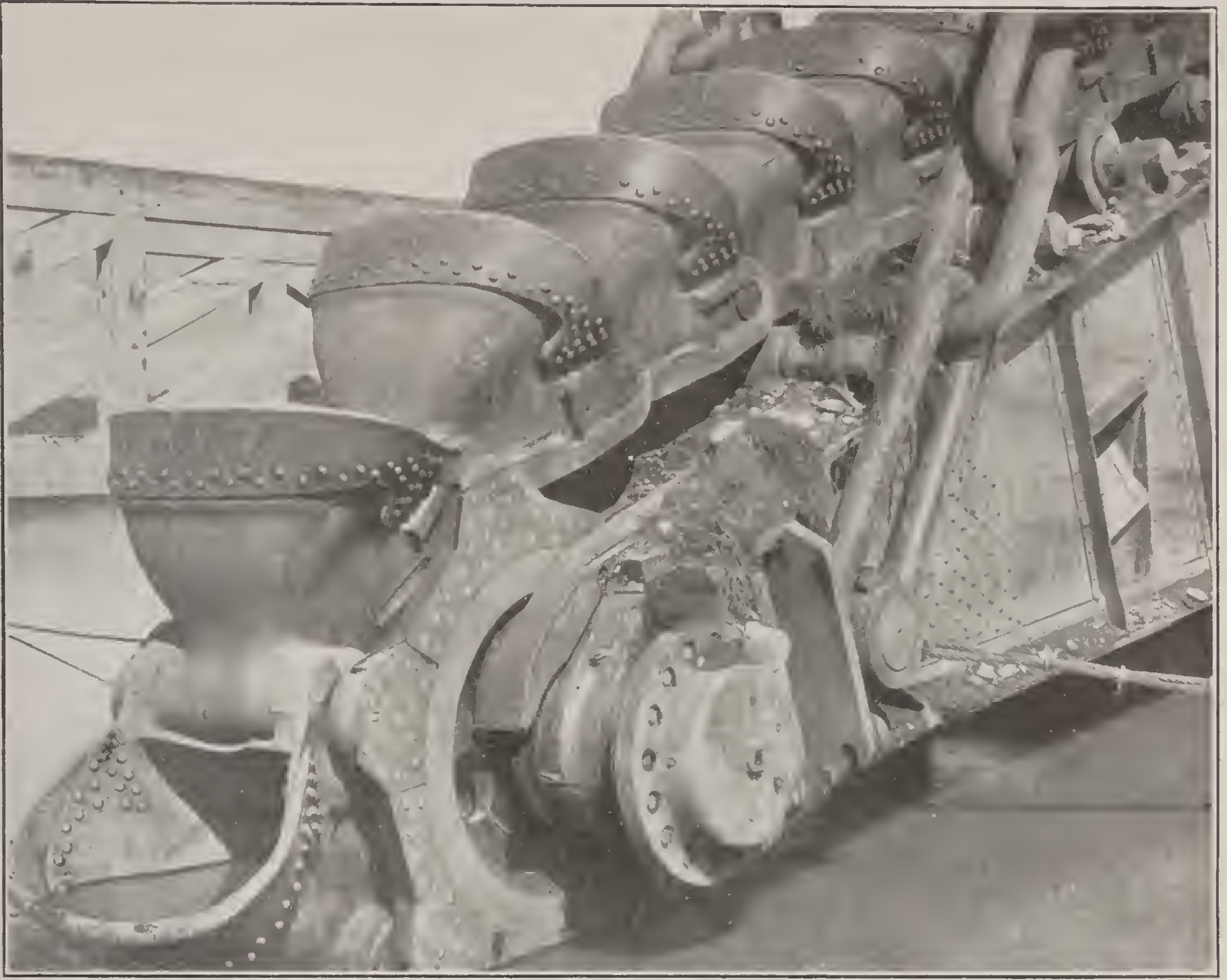
Another type of lower-tumbler bearings worked out in the Ruby district has proved successful and is in general use there. The main steel bearing for the lower-tumbler shaft is fastened to the end of the ladder with two large bolts. The bearing is lined with a circular iron bushing to take the wear. On the outside end of the bearing a large steel cap is bolted with packing between the cap and the bearing; on the inside next to the tumbler a steel ring with



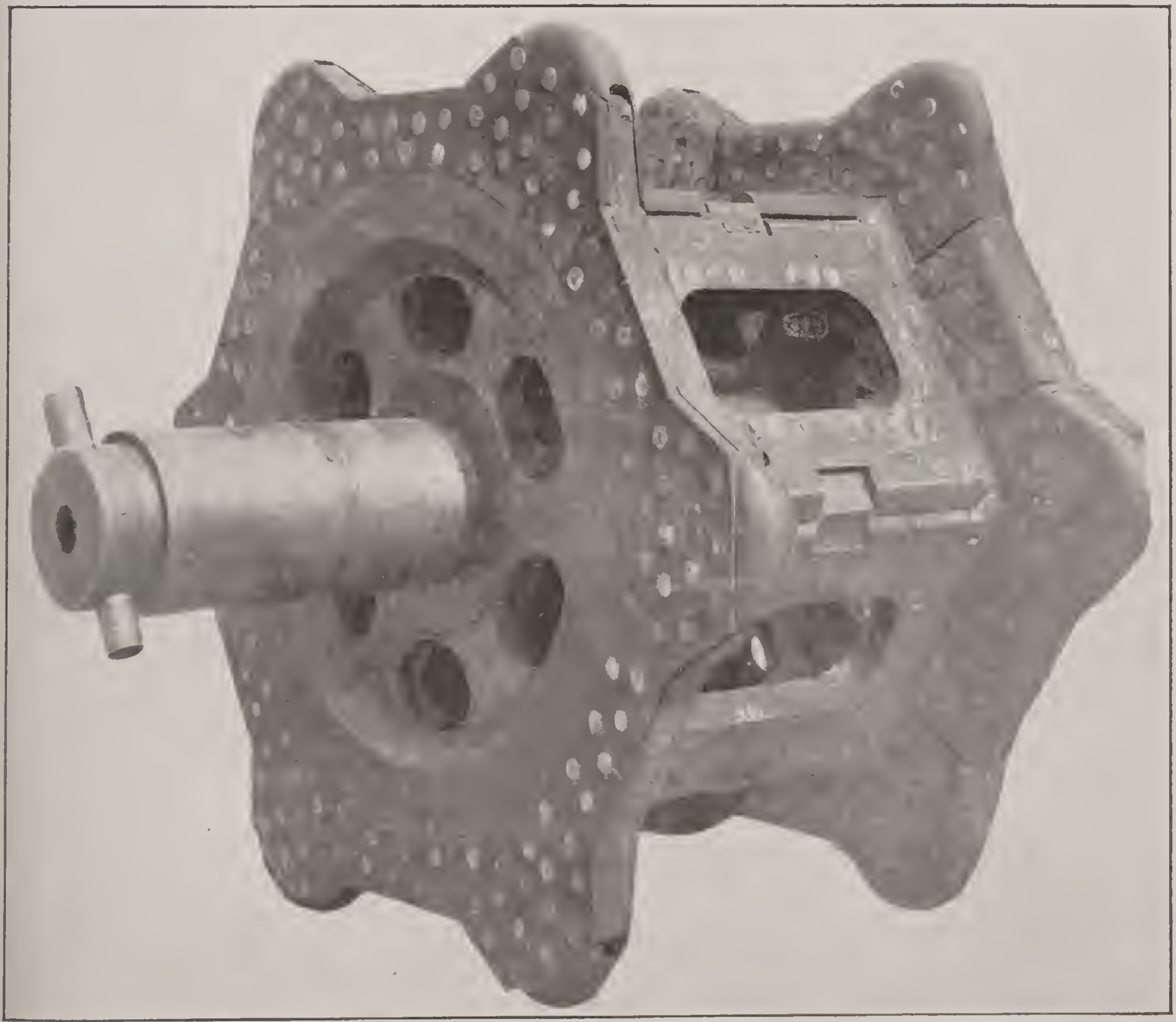
A. SIX-FOOT BUCKET OF GUAMOS DREDGE.

B. ROUND LOWER TUMBLER; ONE-PIECE PATTERN; MANGANESE STEEL.

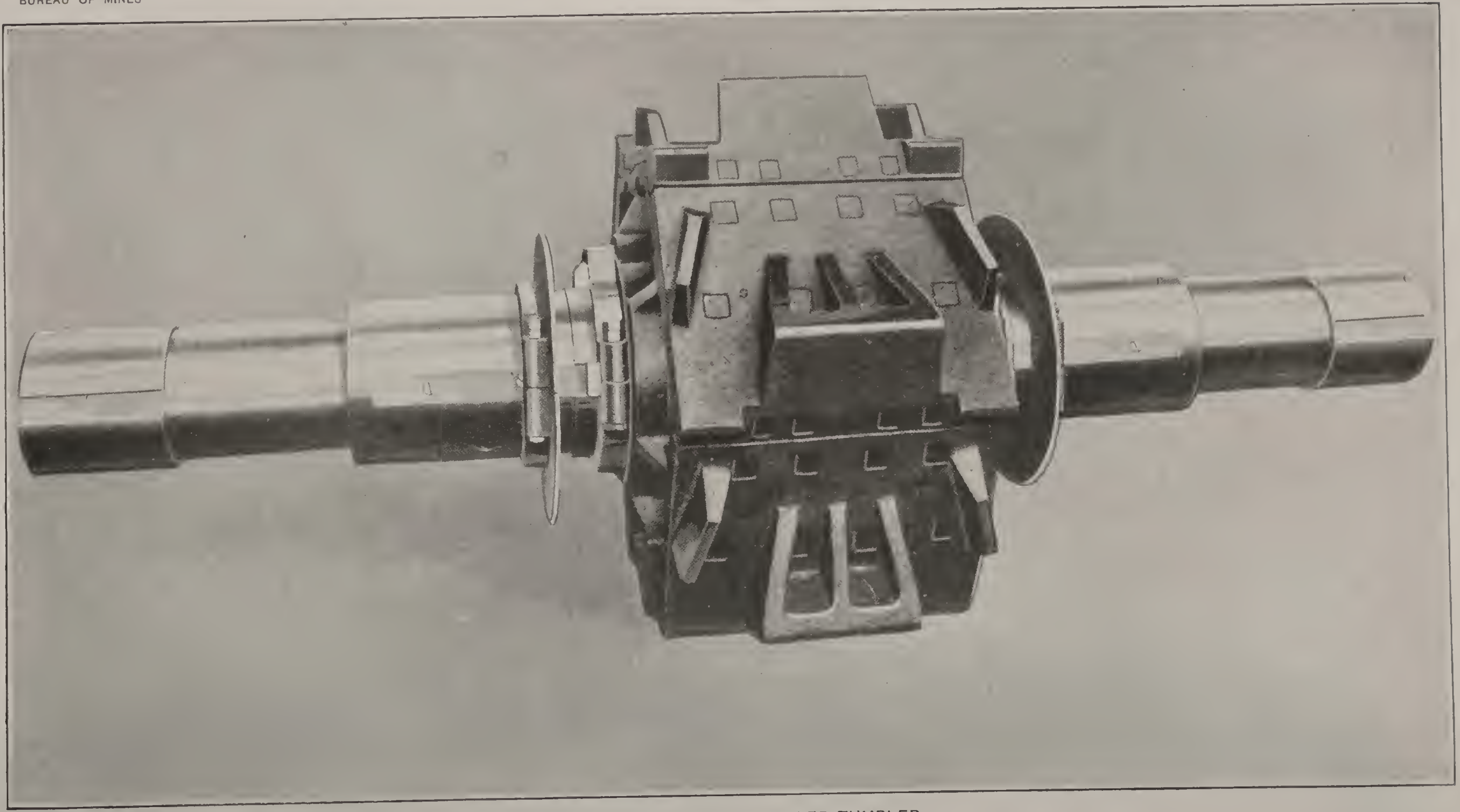
C. HEXAGONAL TUMBLER MADE ROUND BY ADDING CAST-STEEL SEGMENTS.



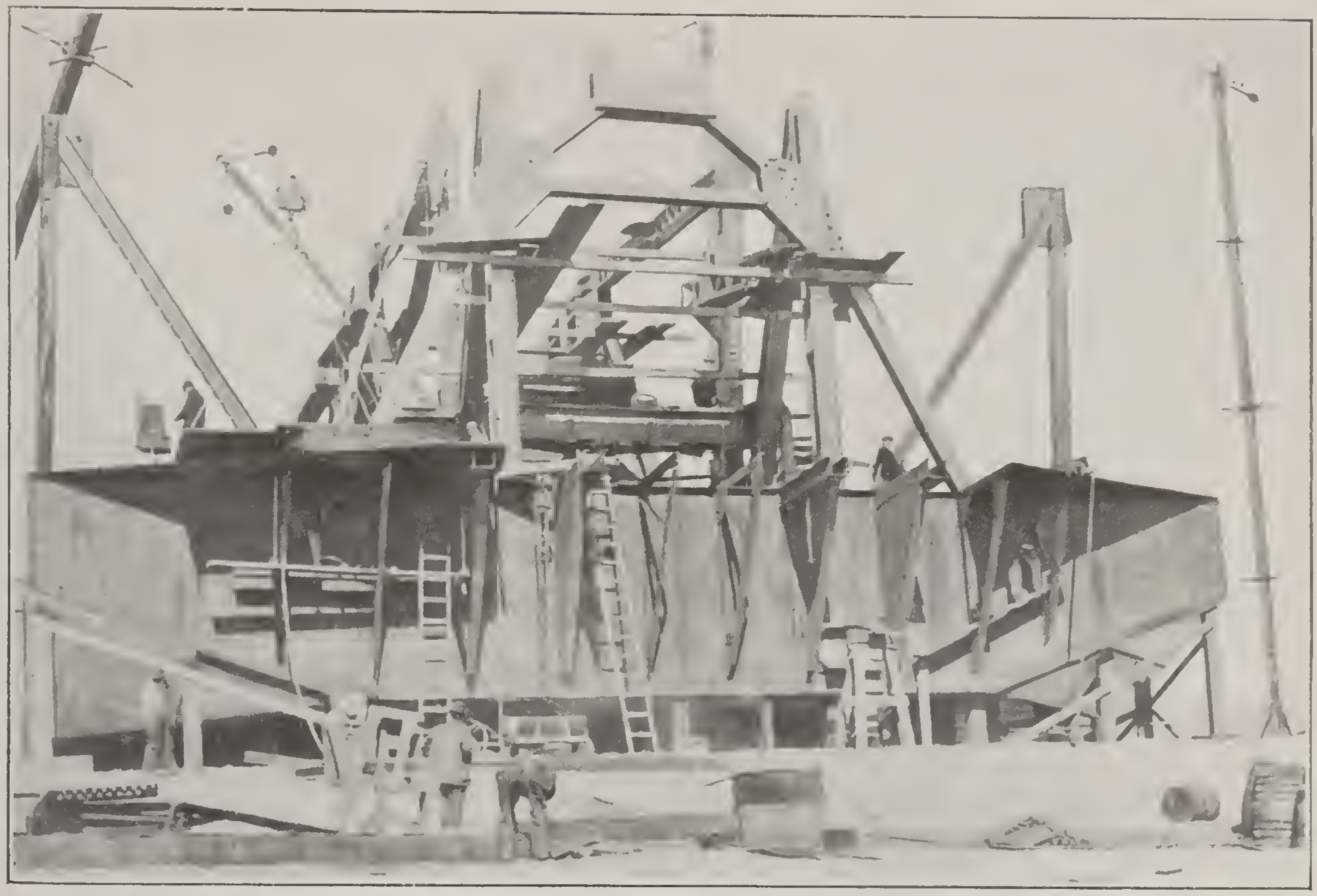
A. BEARING OF LOWER TUMBLER ON DREDGE AT RUBY, MONT.



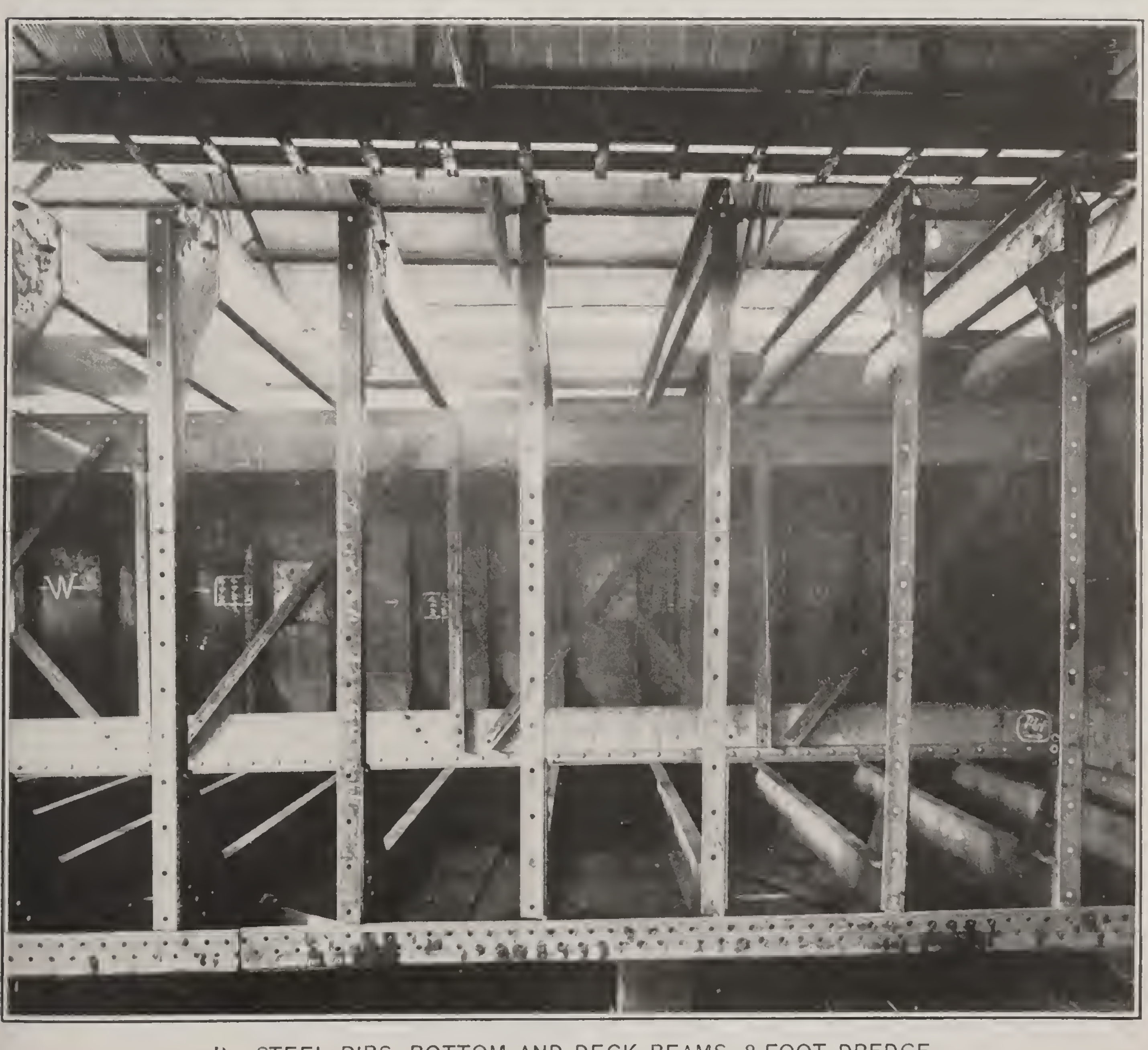
B. HEXAGONAL LOWER TUMBLER WITH DEVICE TO PREVENT LADDER FROM SPREADING.



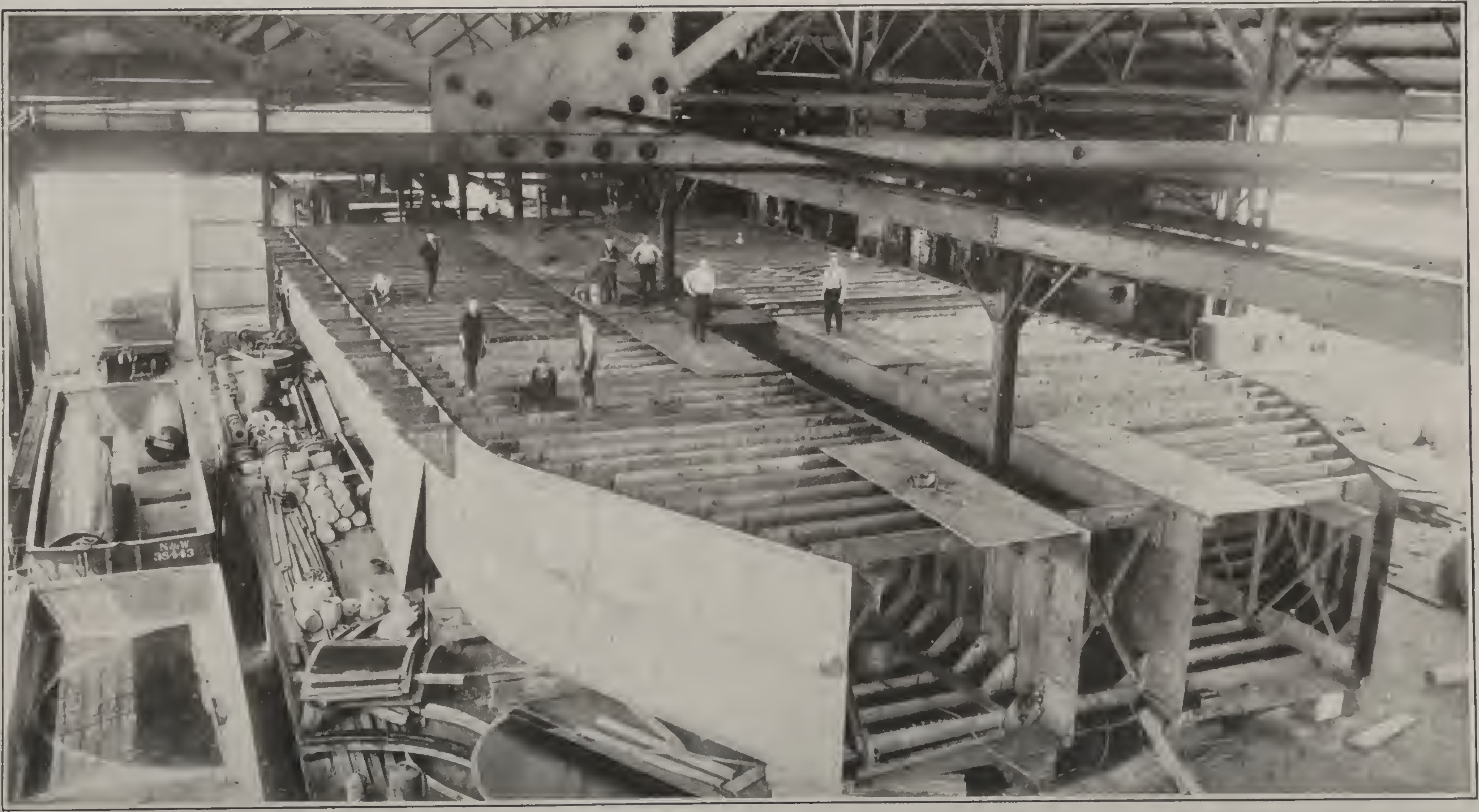
CUSHION-PLATE TYPE OF UPPER TUMBLER.



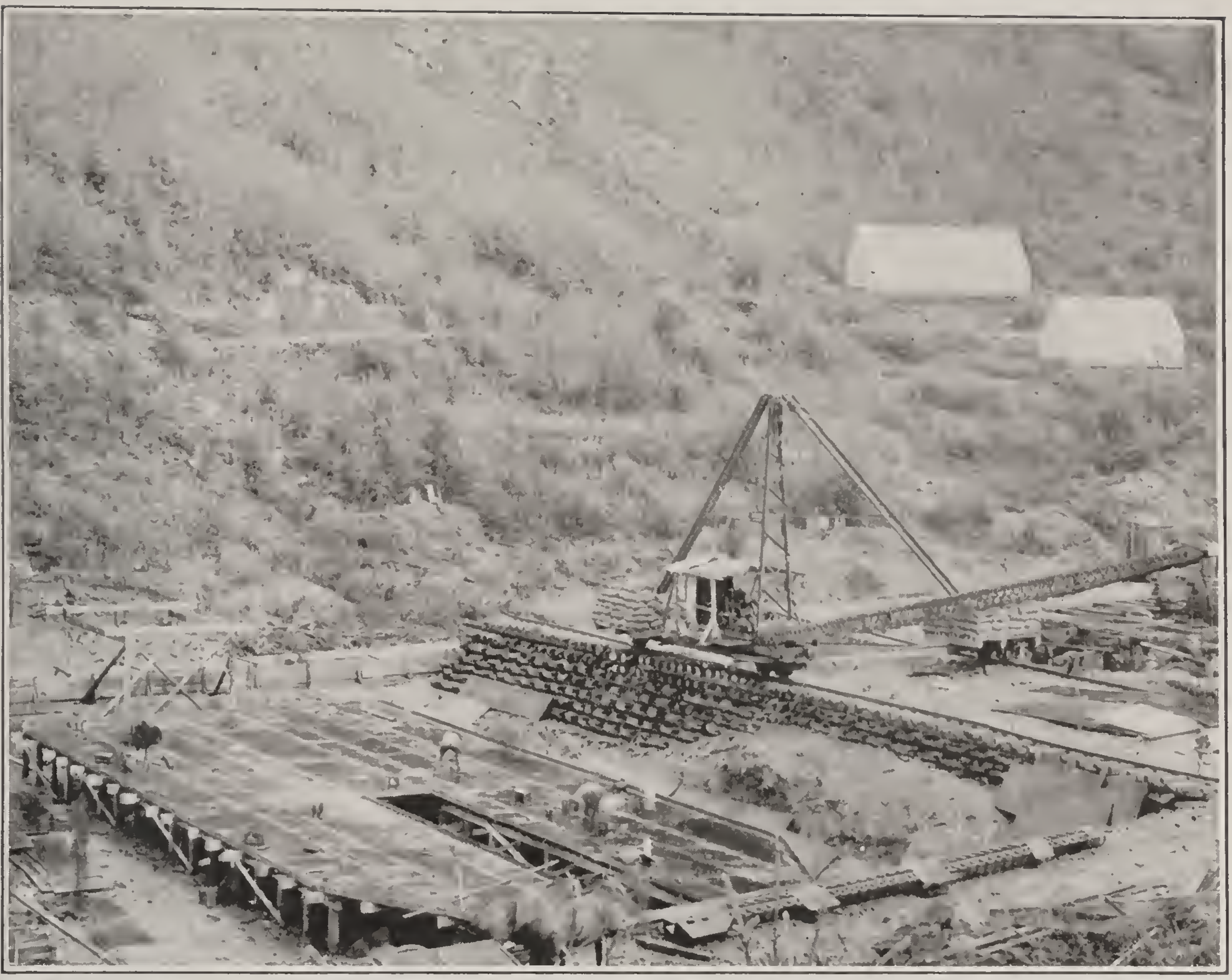
A. STEEL HULL WITH BEVELED STERN, YUBA NO. 15 DREDGE.



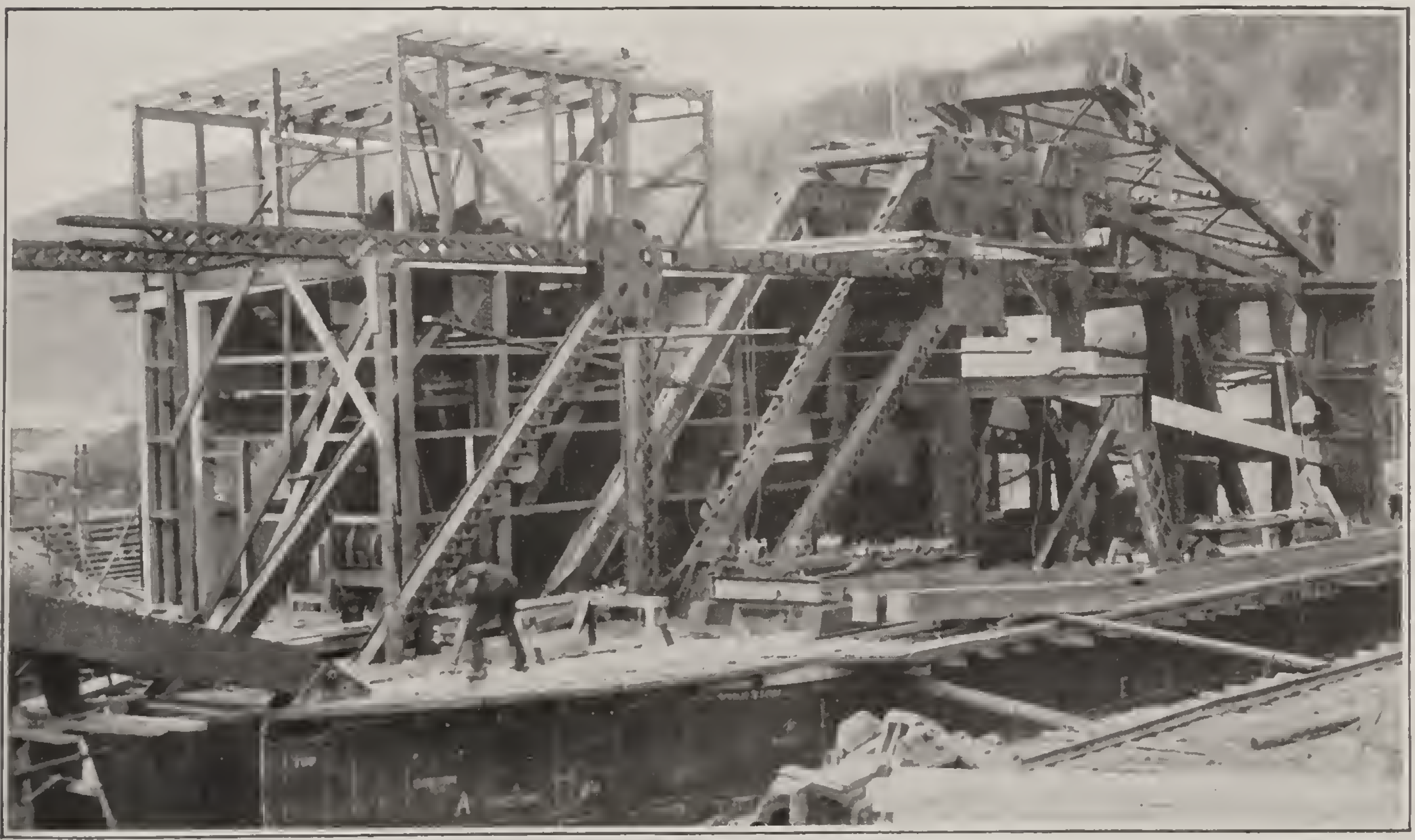
B. STEEL RIBS, BOTTOM AND DECK BEAMS, 8-FOOT DREDGE.



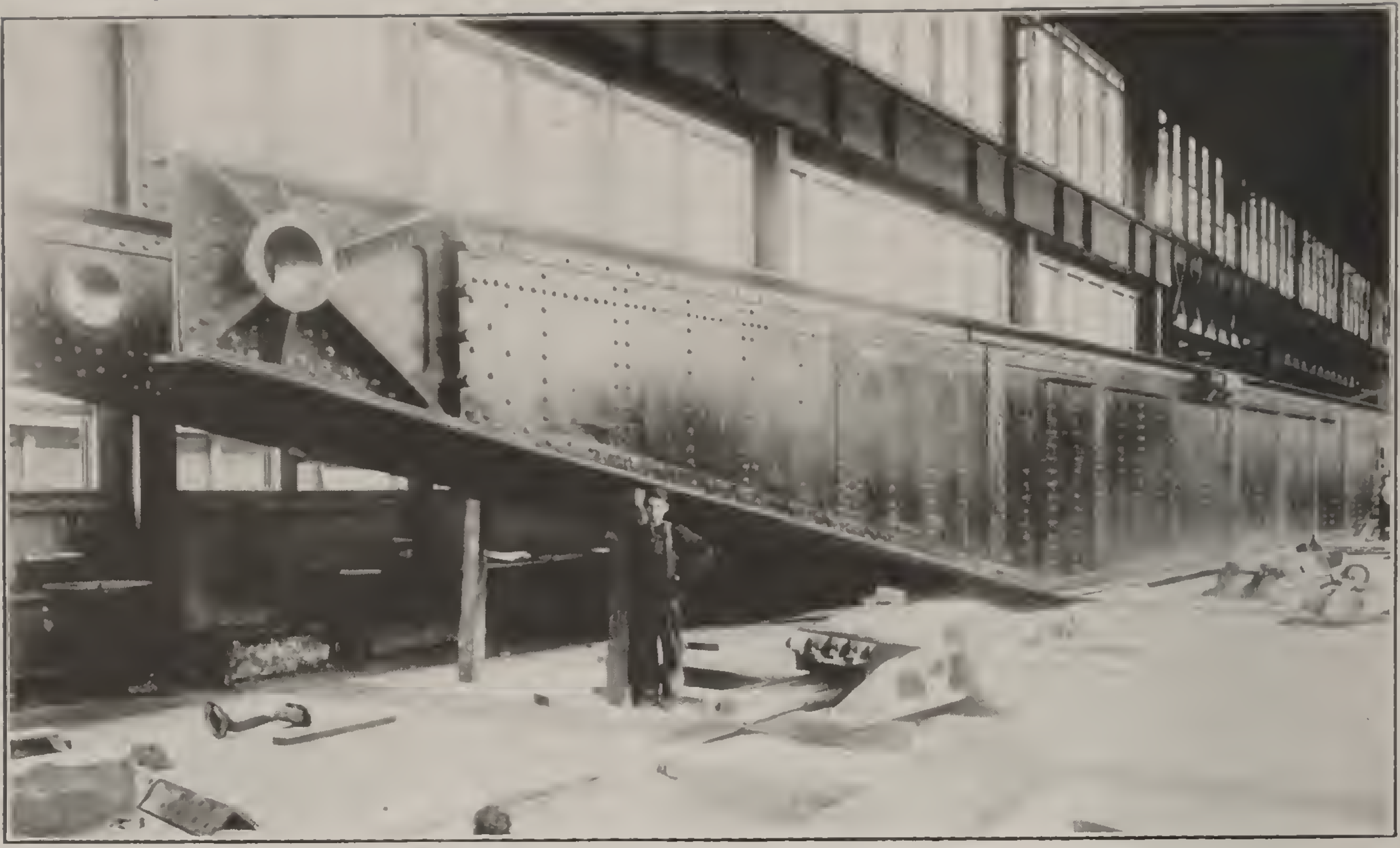
STEEL HULL UNDER CONSTRUCTION.



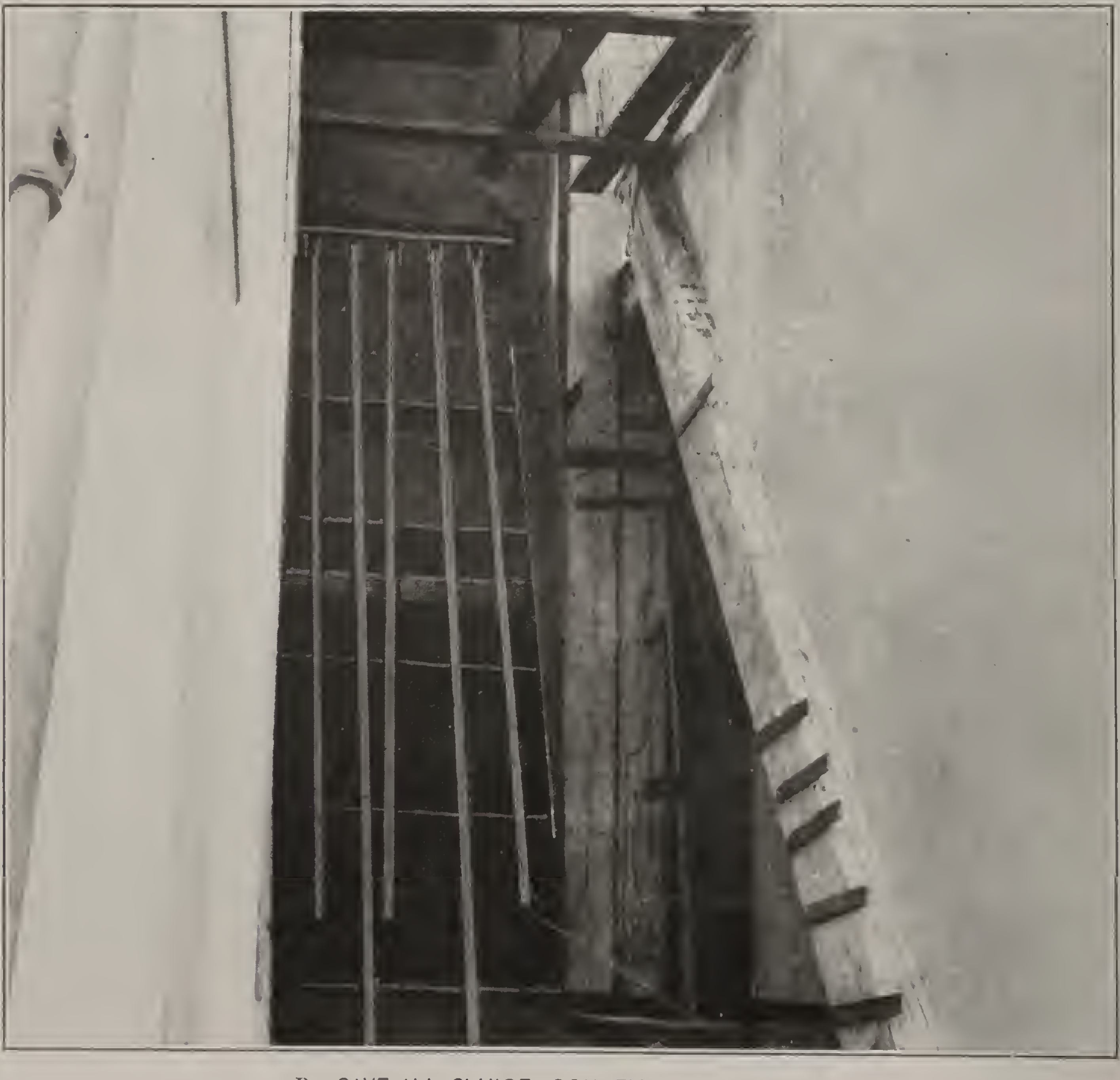
A. YUKON NO. 9 DREDGE UNDER CONSTRUCTION.



B. SUPERSTRUCTURE OF A STEEL DREDGE,



A. PLATE-GIRDER DIGGING LADDER.



B. SAVE-ALL SLUICE, CONREY NO. 2 DREDGE.

The most desirable height of the bow gantry depends on the length of the ladder (Pl. XXXVII, *A*). It is necessary to make the gantry high enough so that the ladder lines will not pull at too small an angle when the ladder is at its highest position. The bracing of the gantry must be arranged so the ladder can be raised to such a height that the lower tumbler can be rested on the bank in front of the dredge during repairs, for otherwise it is necessary to dig away the bank. The present practice is to build the bow gantry as a unit and to set it in bases that allow pivoting. This is a great convenience during erecting as the lower end can be placed in the bracket and the gantry swung into place. The old method was to build the posts into the hull.

The stern gantry supports the outer end of the stacker and the spuds. It should be high enough to allow the spuds to be raised clear of the lower supports so that they can be taken out without dismantling the supports.

LADDER ROLLERS AND BEARINGS.

Ladder rollers except for a few minor details, have become standardized. Formerly three hubs were used—one on each end and one in the middle, the shaft being pressed in. The intent was to have a press fit at the three hubs, but owing to the fact that it was practically impossible to apply calipers to the center hub accurately this center bearing is no longer favored by most dredge makers, as it adds nothing to the strength of either the roller or the shaft. Many dredge makers use manganese rollers, which are well suited to this service. The only objection to a manganese roller is that if it becomes loose on the shaft the bore wears oblong and it can not be used with another shaft. Nickel-chrome steel is the next best material for roller construction, and next to that high-carbon steel.

The ladder-roller shafts wear out rapidly and some shafts are being used with manganese or nickel-chrome sleeves. The design of the bearings determines the life of the shaft. The method of suspending the bearing on lugs that extend below the bearing (Pl. XXXVIII, *A*) has not been found the best practice on account of the greater movement and consequent wear when this type is used as compared to the method of pivoting the bearing on the center line of the shaft (Pl. XXXVIII, *B*).

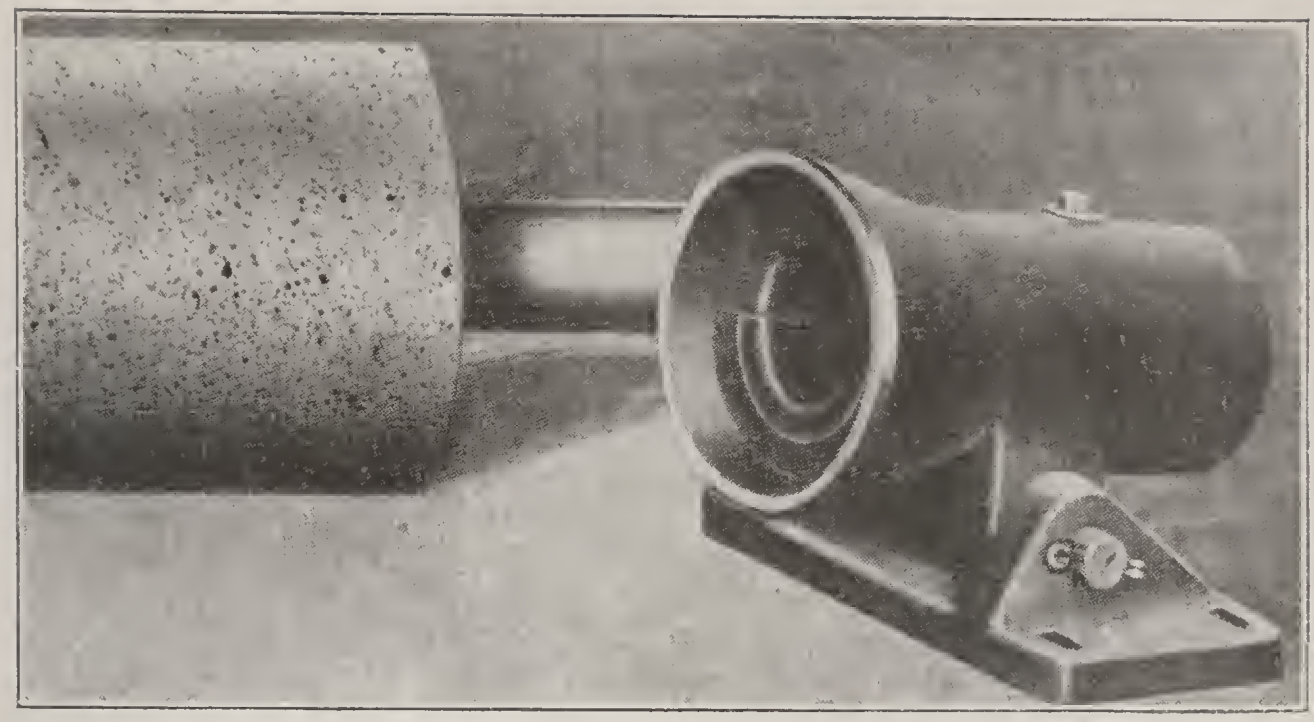
Ladder-roller bearings are about the only parts of the modern dredge made of cast iron, and the cast-iron bearings are being replaced with steel.

MAIN HOPPER, SAVE-ALL, AND BUCKET IDLERS.

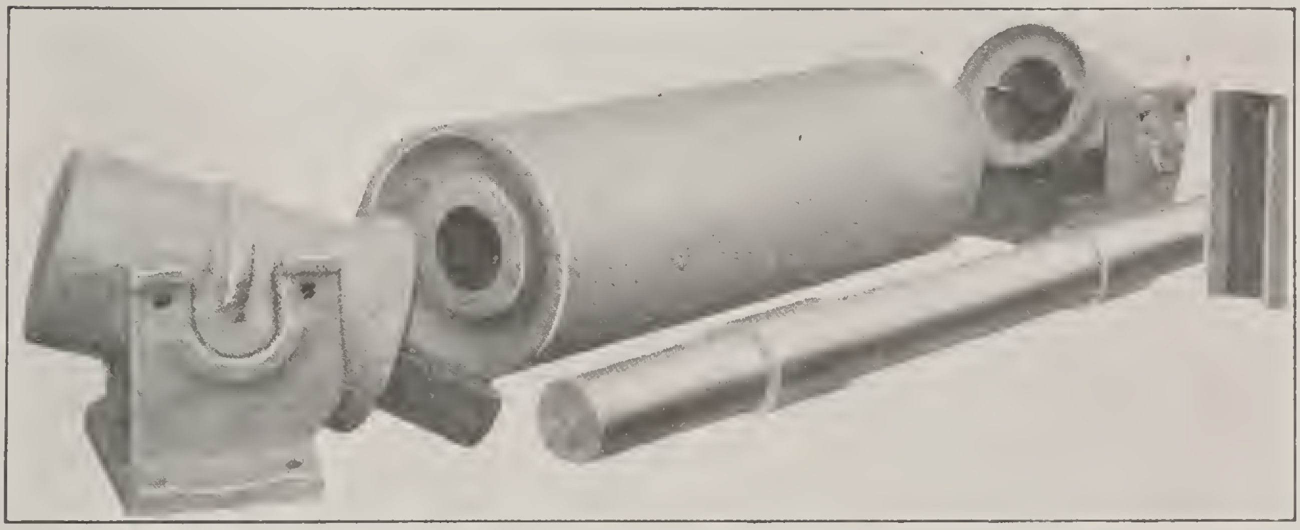
For several reasons it is desirable to have the upper tumbler no higher above the deck than necessary. Such construction not only lowers the center of gravity and therefore increases the stability of the dredge, but it keeps down the weight. Of greater importance is the waste in power in raising the gravel the extra distance necessitated by an unduly high upper tumbler, and in hoisting the additional weight of the ladder and in the bucket line.

The distance between the upper tumbler and the deck can be kept to a minimum only if the main hopper into which the material is dumped from the buckets is designed to avoid a long drop. The hopper must also be designed to prevent excessive spilling. In general shape the main hopper is like a box with the top removed and one side cut away to within a short distance of the bottom. This distance is just great enough to keep the material from running over and should allow minimum clearance of the buckets when they are digging the maximum depth. The spout leading from the hopper to the screen is set about 6 to 8 inches above the bottom so that the hopper fills with gravel to that distance and there is no wear on the bottom. The side removed should be just wide enough for the buckets to pass through. Chutes are placed on each side to carry into the hopper material spilled over the sides. Water jets should play against the sides of the buckets and on the inside so that no material will be carried back to the pond. The inside of the hopper should be lined with manganese steel bars so designed that when the first set of bucket pins need replacement the worn pins can be used to line the hopper. Worn bucket pins can be used to great advantage here.

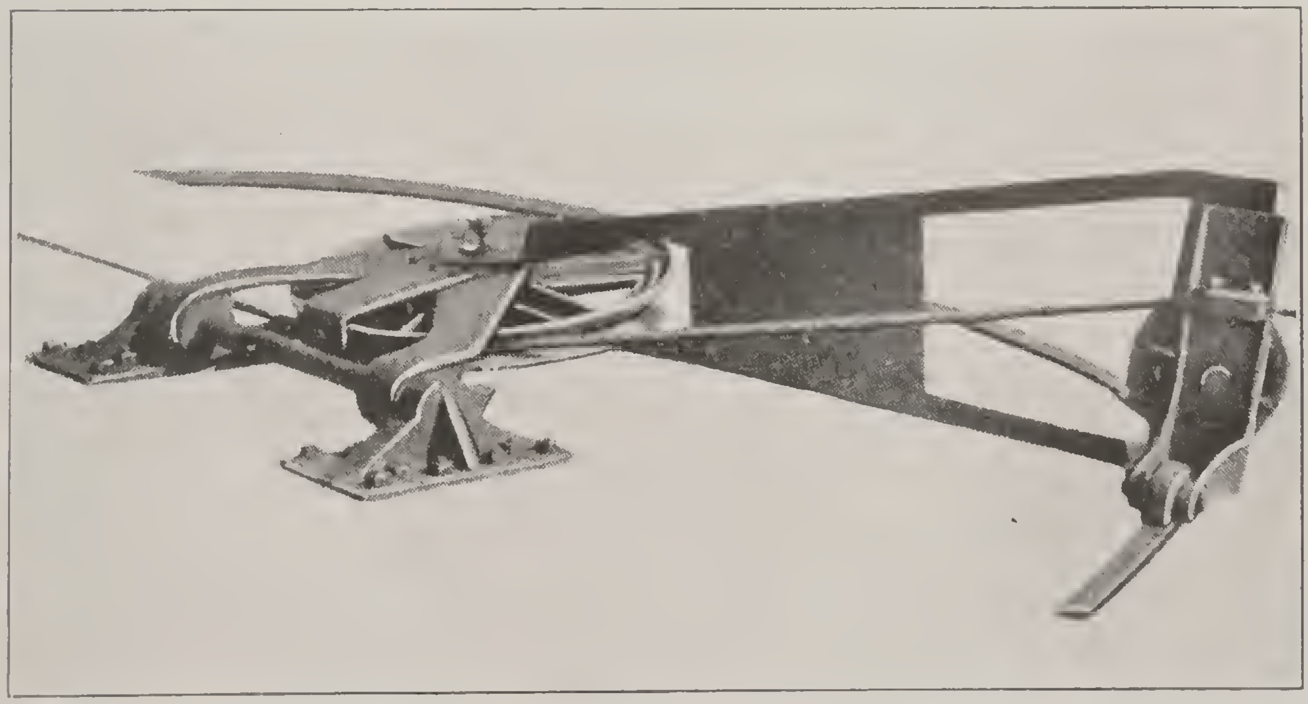
The save-all sluices (Pl. XXXVII, *B*) should have a greater pitch than the tables owing to larger wash. It is difficult to make them so that they are easily accessible for cleaning. A grizzly (Pl. XXXIX, *A*) with about 2-inch openings should be placed above them to keep out the larger rock. The most important point in connection with the grizzly is to make it so that it can be easily repaired. With the old type of bars repair was very difficult, as the bars were held together by rods running through holes in the bars with a spacer between. This grizzly was in one piece and owing to the construction at this part of the dredge it was usually necessary to run the rods through the large timbers supporting the main drive, building them into the dredge. The removal of the rods was difficult; it was necessary to handle the bars as a unit, and some of the grizzlies were 8 feet wide and 14 feet long and weighed over 6,000 pounds. The best practice is to cast a rack in which the bars can be set, each bar being removable, as the wear is limited to a small area and it is not necessary to throw the whole grizzly away because this area becomes worn. When one end of the bar is worn it can be turned end for end and then moved to the edge where the service is not severe. This type of grizzly, which is shown in Plate XXXIX, *A*, will wear many times as long as the old type.



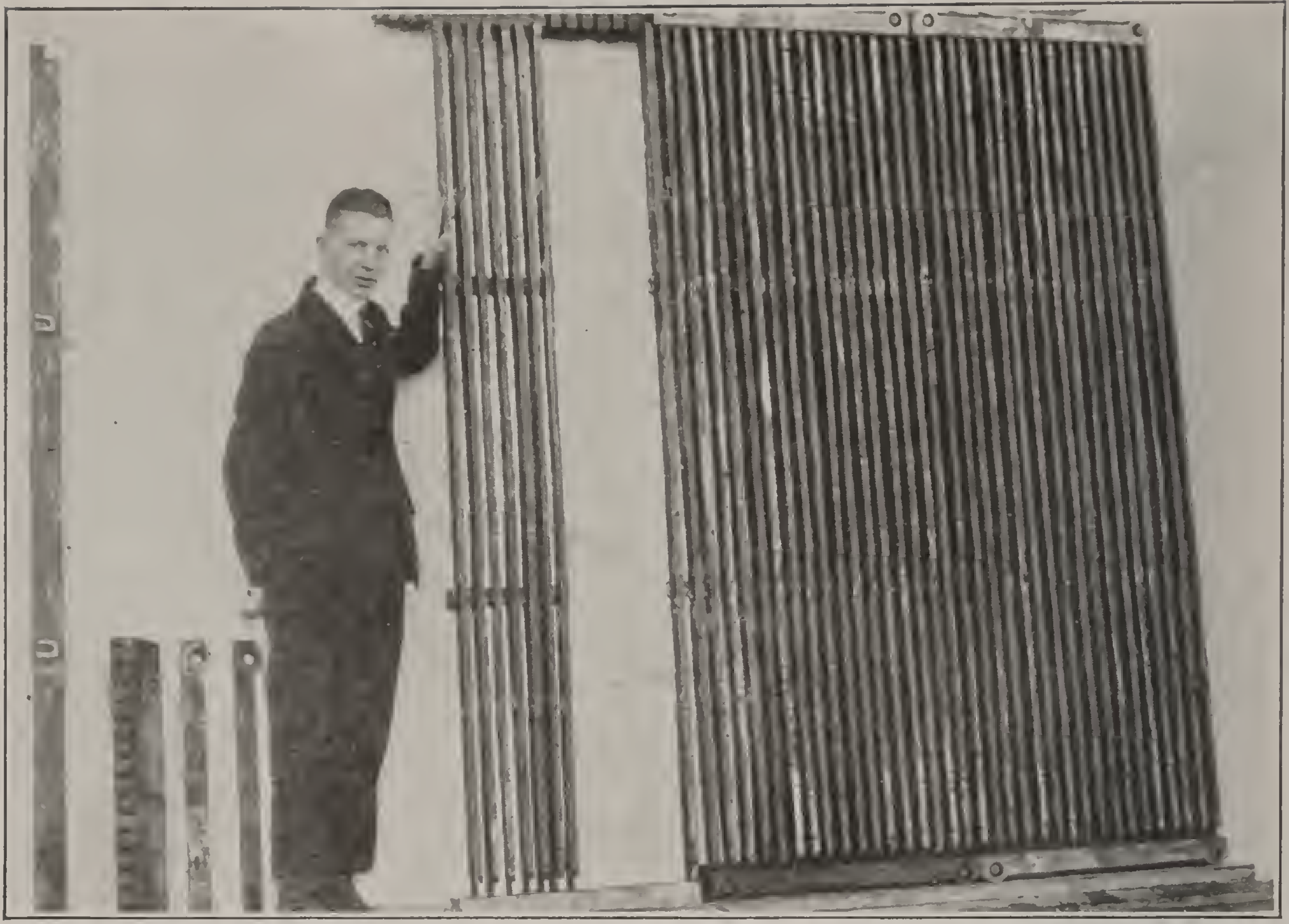
A. LADDER-ROLLER BEARING SUSPENDED ON LUGS.



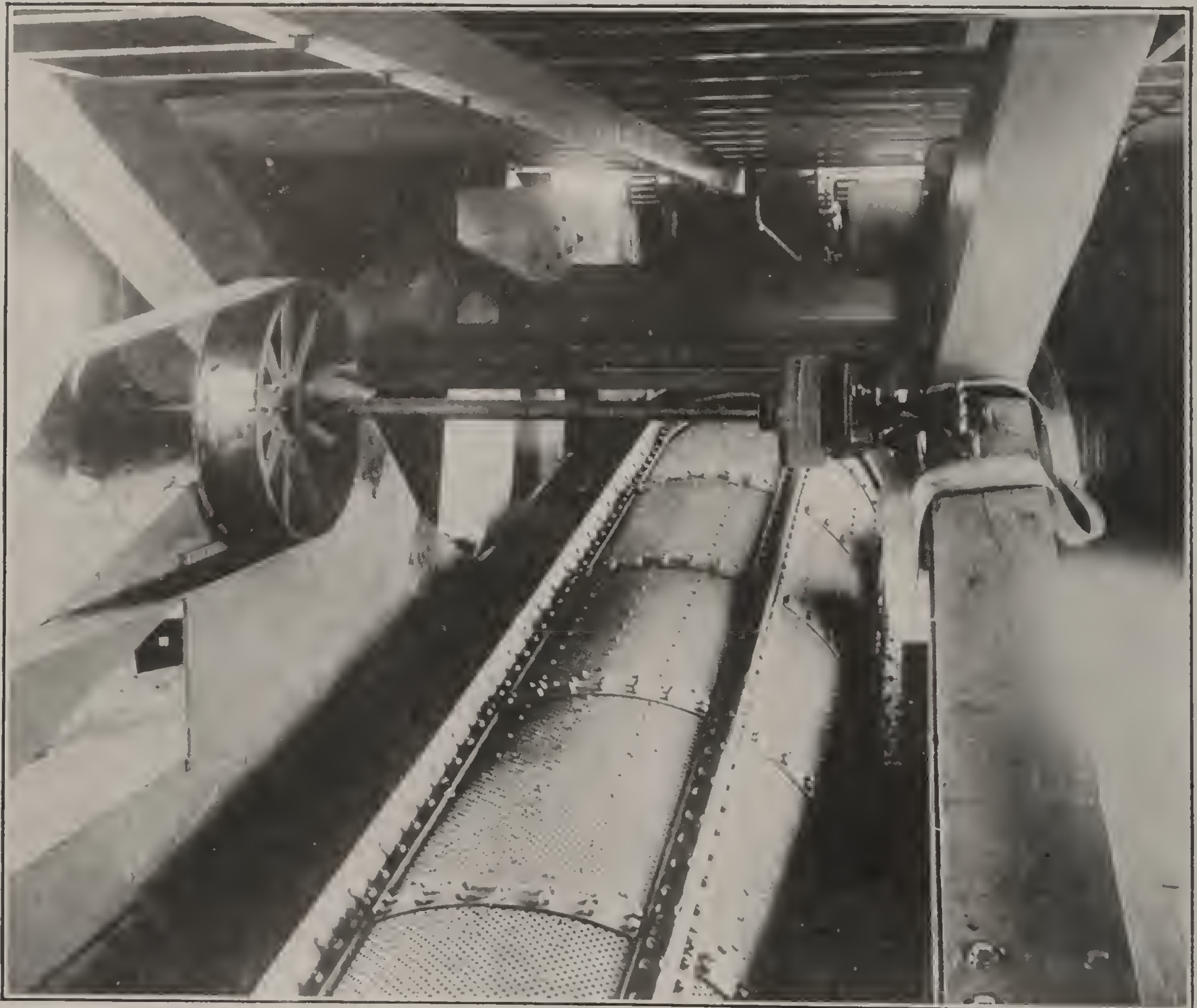
B. LADDER-ROLLER BEARING, PIVOTED TYPE.



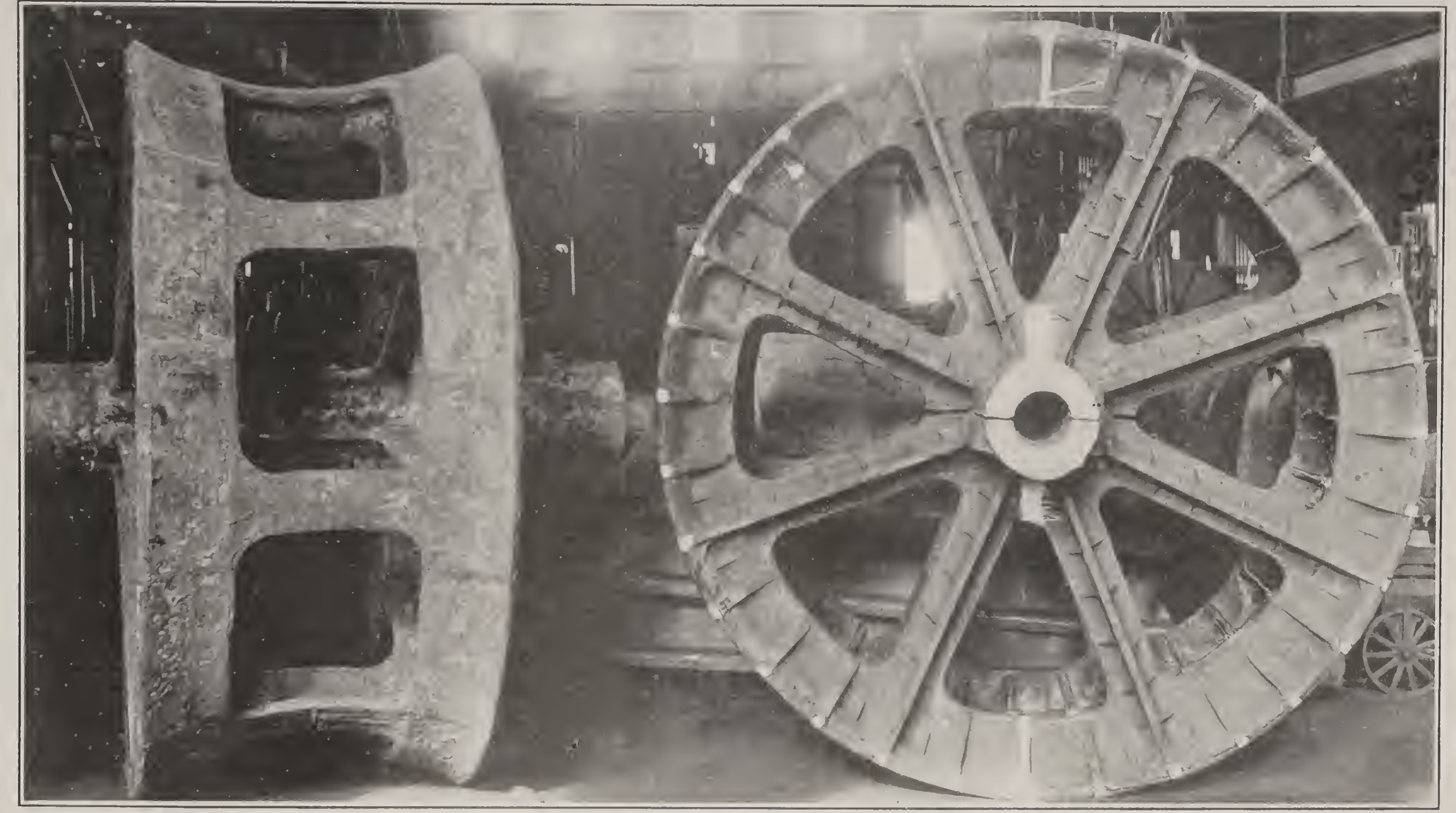
C. SHEAVE FOR BOWLINE.



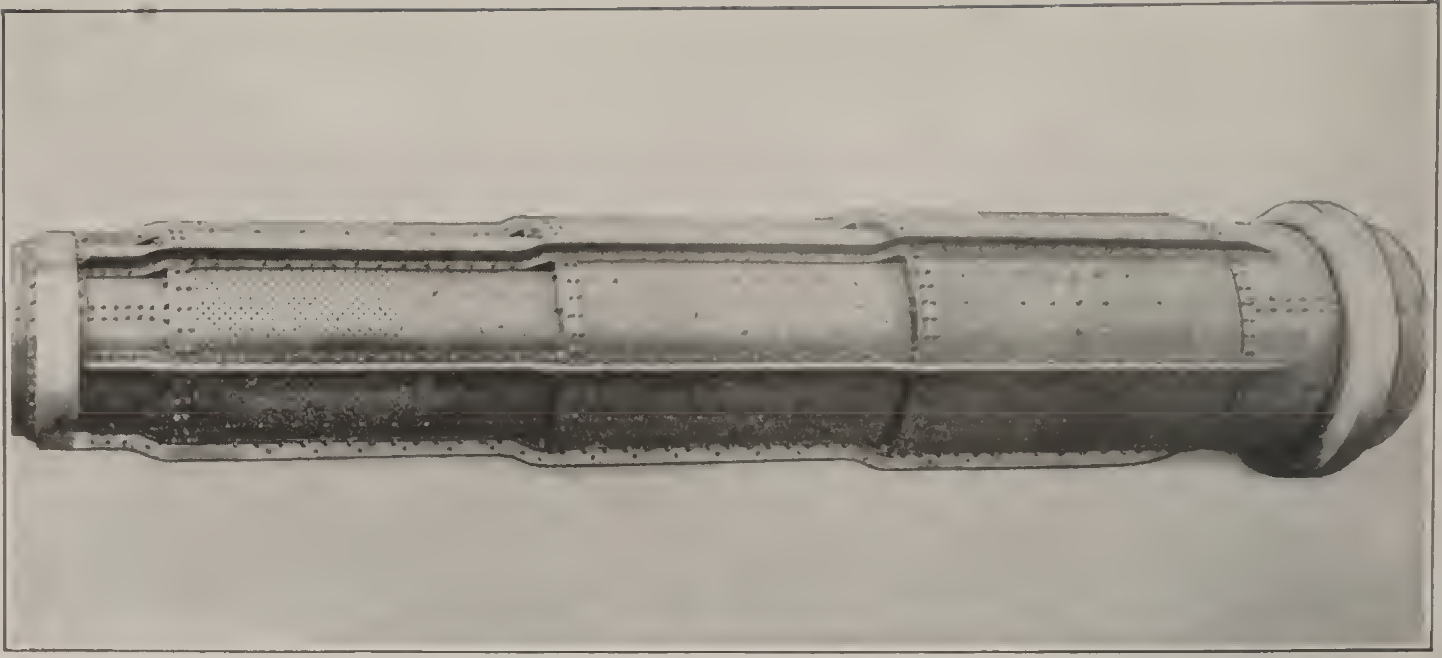
A. GRIZZLY OVER SAVE-ALL SLUICE; HAS CAST RACK FOR HOLDING MANGANESE-STEEL BARS.



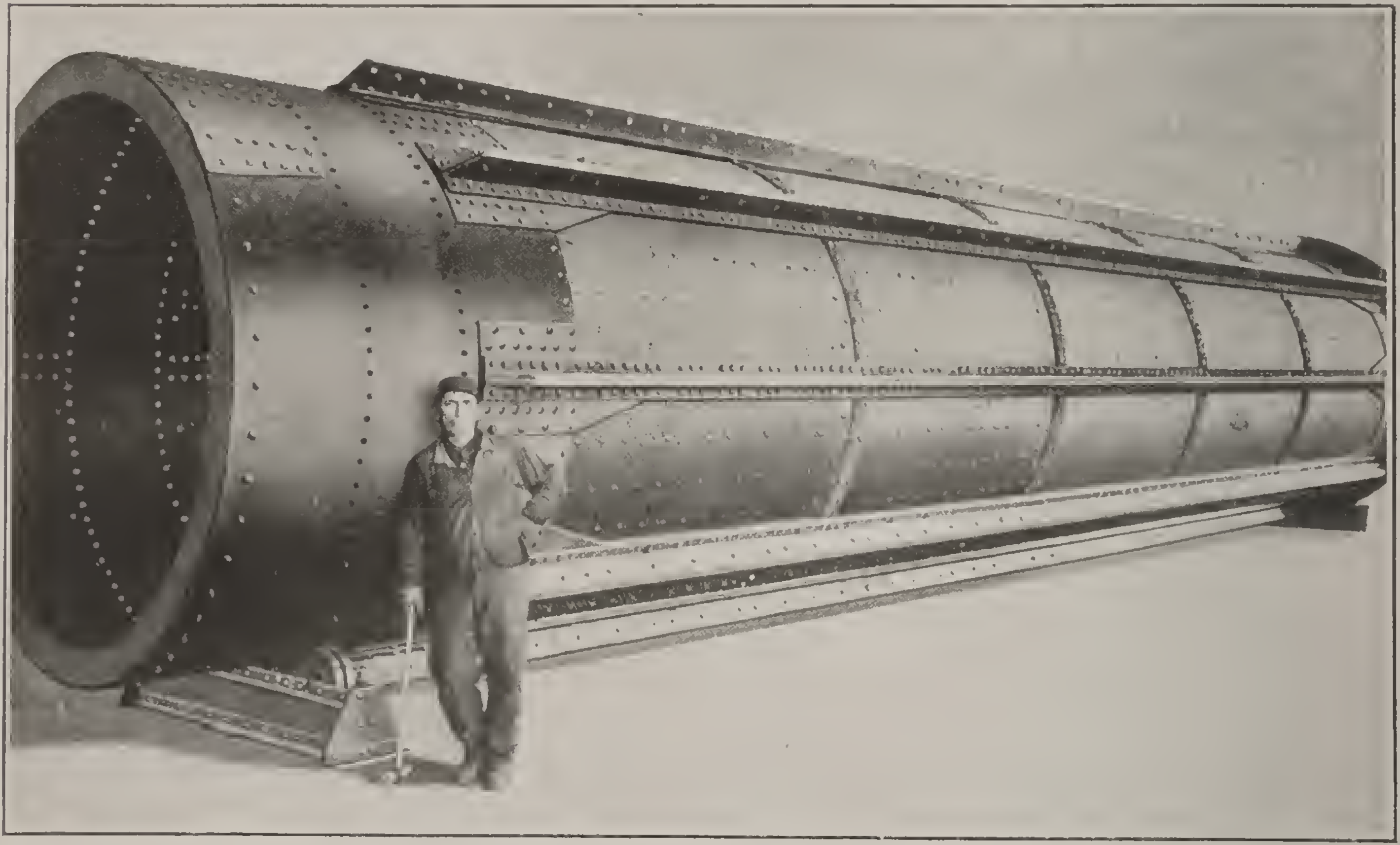
B. SCREEN ON UNION DREDGE SHOWING DRIVE AT UPPER END.



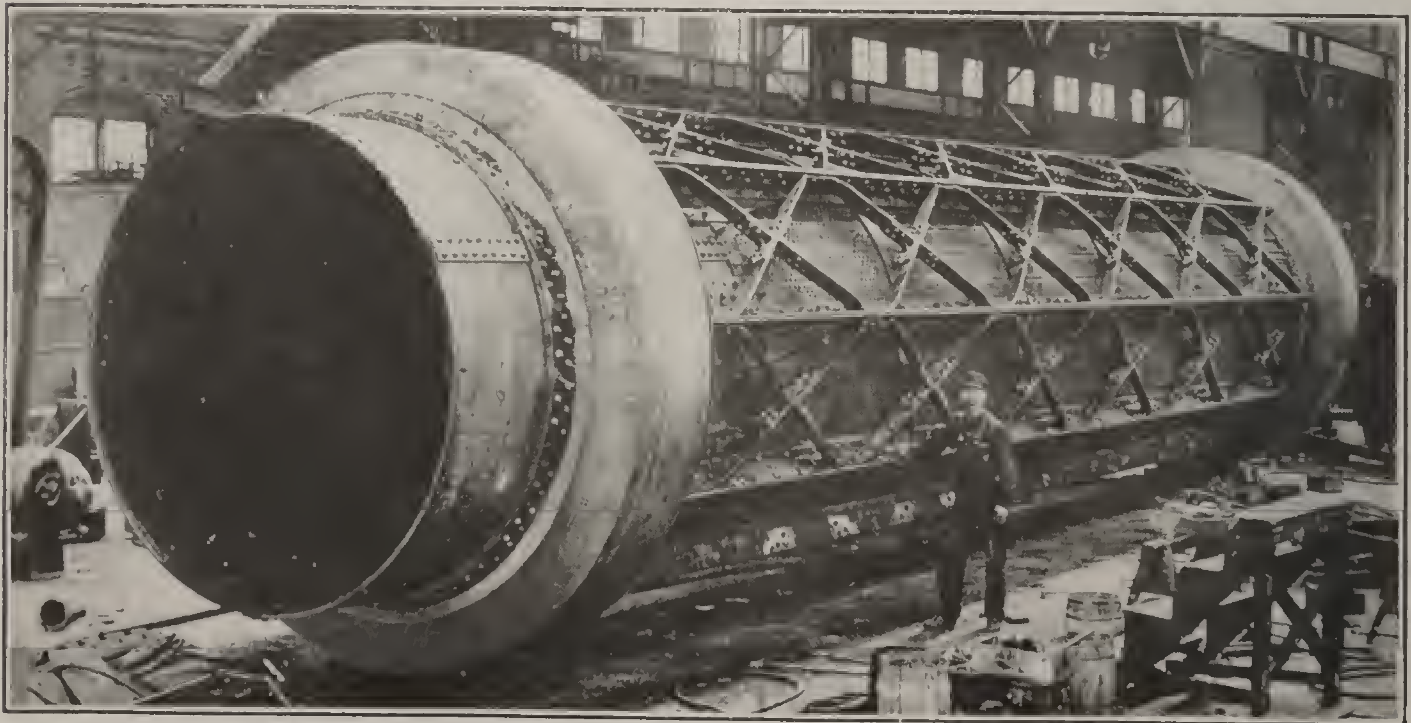
ONE-PIECE BUCKET IDLER.



A. REVOLVING SCREEN, SEGMENTAL STEP TYPE.



B. CYLINDRICAL SCREEN, 50½ FEET LONG AND 9 FEET IN DIAMETER FOR 15-FOOT DREDGE.



C. SCREEN FOR CONREY NO. 4 DREDGE. NOTE METHOD OF BRACING.

the smaller diameter a trifle larger than the shaft is held by stud bolts in the end of the bearing casting. The smaller diameter of the ring is beveled to take square flax packing, which is compressed against the cast-iron bushing in the bearing and the shaft by tightening up the stud bolts. The cap on the outside of the bearing is large enough to allow a large steel collar, pinned on the end of the tumbler shaft, to turn. This collar has a bearing against the outer end face of the bearing to take the side thrust in digging. Grease is forced into the space surrounding the thrust collar inside of the cap to give perfect lubrication to all wearing parts. The ladder roller bearings are essentially the same as those for the lower tumbler shaft. Each bearing consists of two castings: First, a base bolted to the top of the ladder; second, the bearing proper fastened to the base by means of a horizontal pin to give vertical play to the bearing if the roller gets out of alignment. The bearing proper has only one end opening, through which the roller shaft projects, and after the shaft is in place a ring, similar to the one on the lower tumbler bearing carrying flax packing, is drawn up against the face of the bearing by stud bolts. The outer end of the bearing is part of the casting and has a brass plate fastened to the inside to take any end thrust from the roller. The bearing bushing is a half-circular brass casting set into a recess in the bearing casting to prevent turning.

UPPER TUMBLER.

As yet no type of upper tumbler has been designed that will take care of the increase in pitch of buckets. The most satisfactory device yet tried is to have easily removable wearing plates made of different thickness and to build up the tumbler as the pitch increases. Formerly the general practice was to use a plain hexagonal tumbler body and to have six cushion plates that covered the entire face of the tumbler. The cushion plates included the flanges or side lugs, as well as the heel. Later practice was to use an additional wearing plate on the surface of the cushion plate that did the driving (see Pl. XXI). This was called the heel plate and it is this plate that is made in a series of thicknesses. Where long heavy bucket lines are necessary, the heel plates under some conditions will wear out in three or four weeks and the cushion plate in about six months. The cushion-plate type of upper tumbler is being eliminated, as the upkeep is too high, tumblers are being designed to take only flat wearing plates.

It seems impossible to get away from wearing plates on upper tumblers, and the consequent shutdowns from loose bolts, etc., because of necessity the wear is confined to one area. In those fields where the service is not severe the problem is not so difficult. The wearing plate covering each face of the tumbler should be in one piece, for when a plate is in several pieces (Pl. XXII, *A*) the pieces work loose and the tumbler face is worn unevenly.

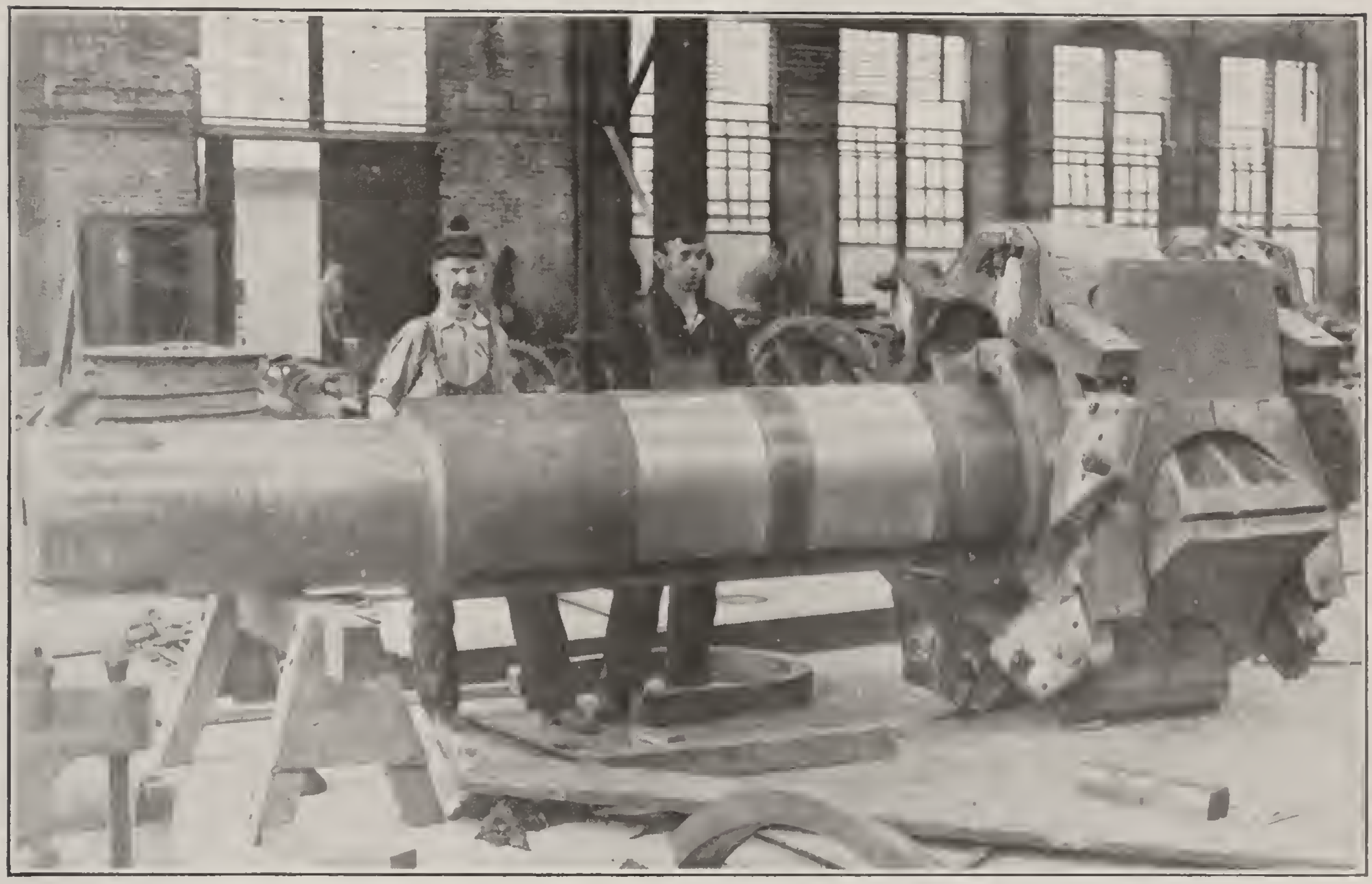
Several of the larger California operators have recently adopted the "one-piece" upper tumbler (Pl. XXII, *B*), which consists of the shaft, tumbler body, and cushion plates, cast in one piece of some special steel. Wearing plates are separate and are of large proportions so that the metal under them will retain a plane surface. The advantage of having the shaft cast with the tumbler is that greater strength is obtained with equal weight and trouble from loose shafts is eliminated. Shrinking the tumbler on the shaft and setting a ring around in a groove, as generally practiced, will hold the tumbler tight but weakens the shaft; hence the groove should not be cut as the shaft starts to crack at this groove. An added element of strength in the one-piece shaft and tumbler is that the ribs in the tumbler body can be extended out on the shaft where the bending moment is greatest and thus be made to strengthen the shaft where it is weakest. When a shaft is not cast with the tumbler, the tumbler should be heated and shrunk on and not pressed on. The relation between bore of tumbler and diameter of shaft should be the same as for the lower tumbler, as mentioned previously.

The increase in the number of sides of the upper tumbler has not kept pace with the increase in size of the dredges. The original dredges in this country followed the practice of those in New Zealand and used the 4-sided tumblers. The number was increased to 5 and then to 6 and has so remained for all sizes. For the 16-cubic-foot dredges there seems to be no reason why 8 sides would not be much superior and this type will no doubt be adopted in the future for dredges digging in ground that can be worked without undue difficulty. Plate XXIII, *A*, shows a method of transporting large tumblers.

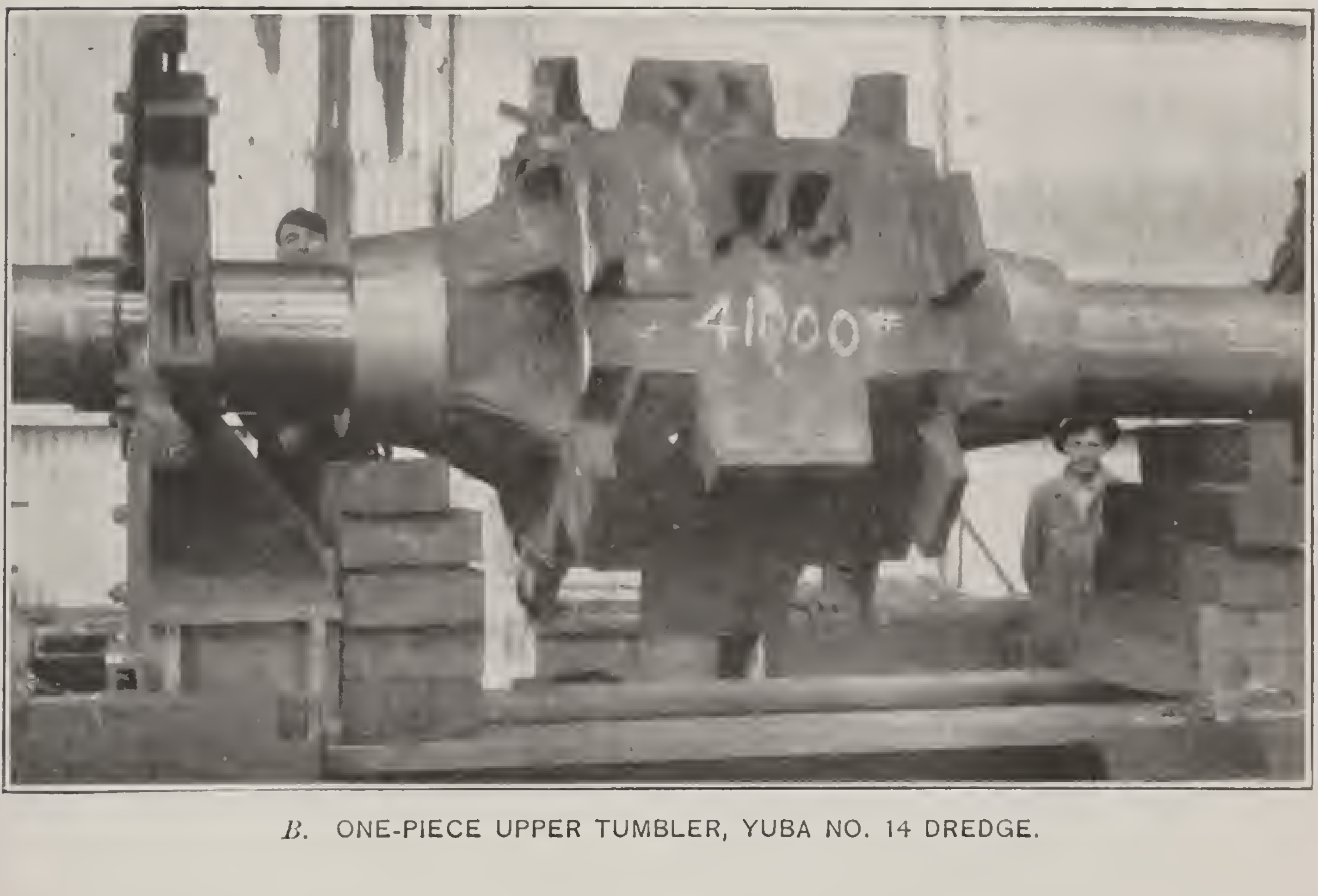
MAIN DRIVE.

The designing of gears (Pl. XXIV) to drive the bucket lines of the smaller dredges—with capacities up to 7 cubic feet—was relatively simple; the gears were made with a sufficiently large factor of safety, so that most gears wore as long as the dredges, but it was necessary to replace the pinions at intervals of two or three years. The "bull gears" were usually 10 to 12 feet in diameter, and the face about 8 to 10 inches wide, and the pitch about 2½ to 3 inches. The gears should be so proportioned that the bucket line will travel about 50 linear feet a minute.

During the year 1916 three 16-foot dredges were designed by different California companies, each one handling the problem in a different way. Two of the companies kept to the same general dimensions as before, the bull gears being 12 feet in diameter and the face 12 inches, but instead of using the ordinary carbon steel for

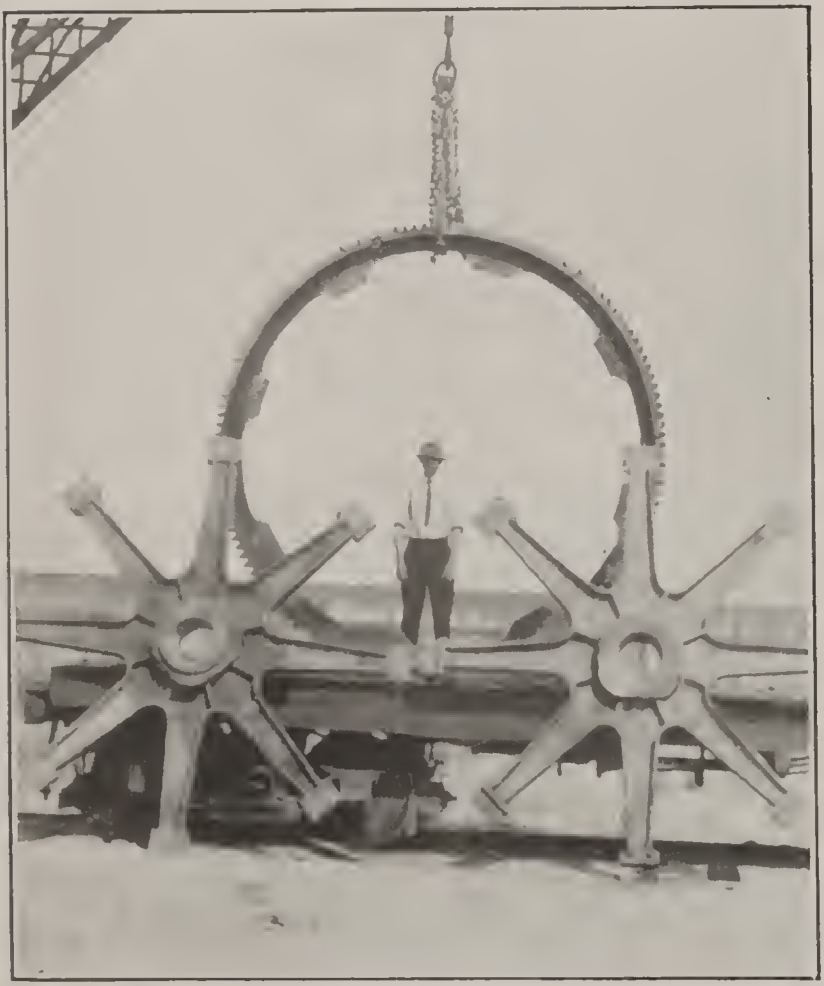


A. HEXAGONAL UPPER TUMBLER WITH THREE-PIECE WEARING PLATES, CONREY NO. 4 DREDGE. ILLUSTRATION SHOWS FITTING OF TUMBLER TO SHAFT.



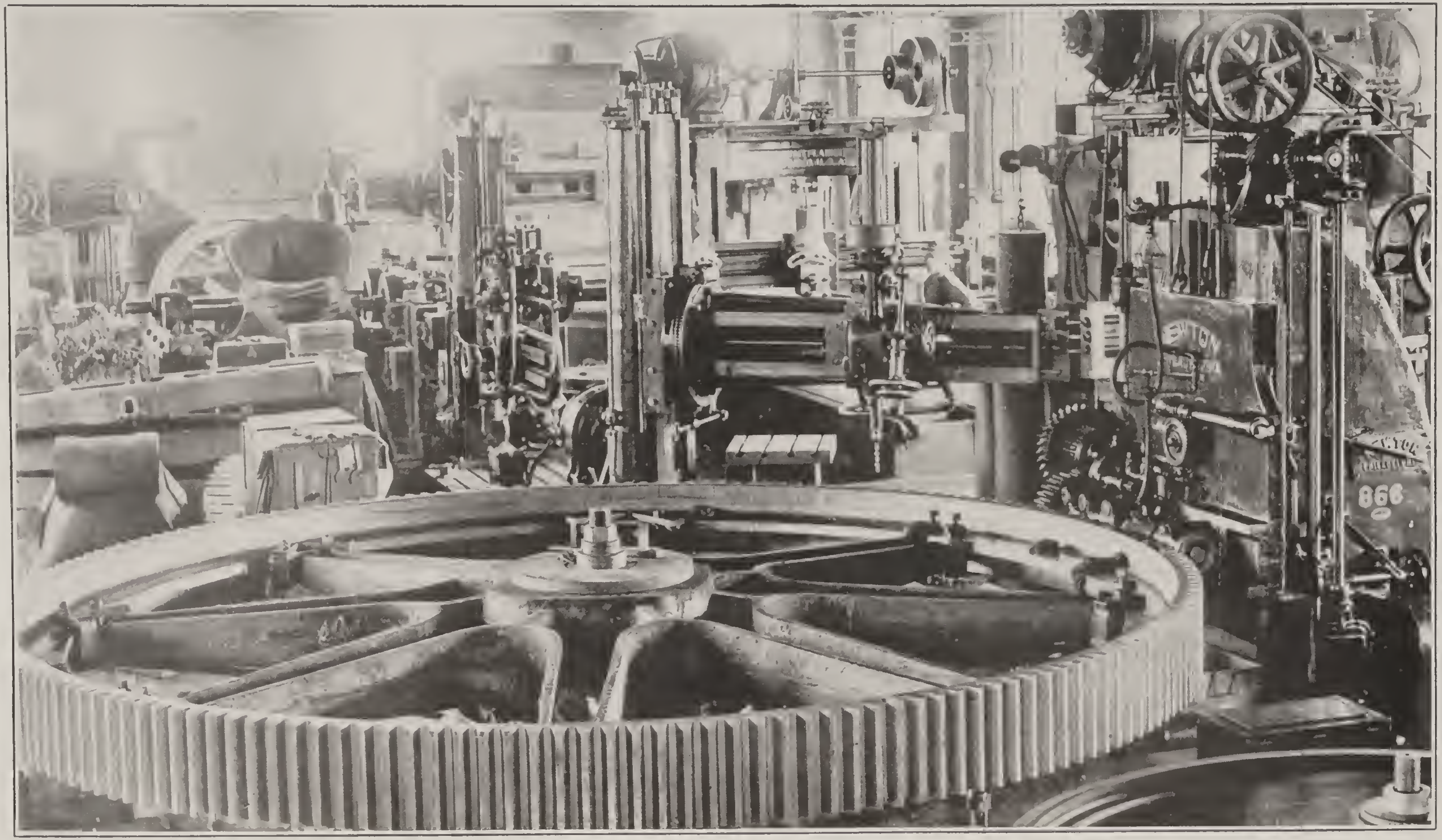
B. ONE-PIECE UPPER TUMBLER, YUBA NO. 14 DREDGE.

A. HAULING A LARGE TUMBLER.

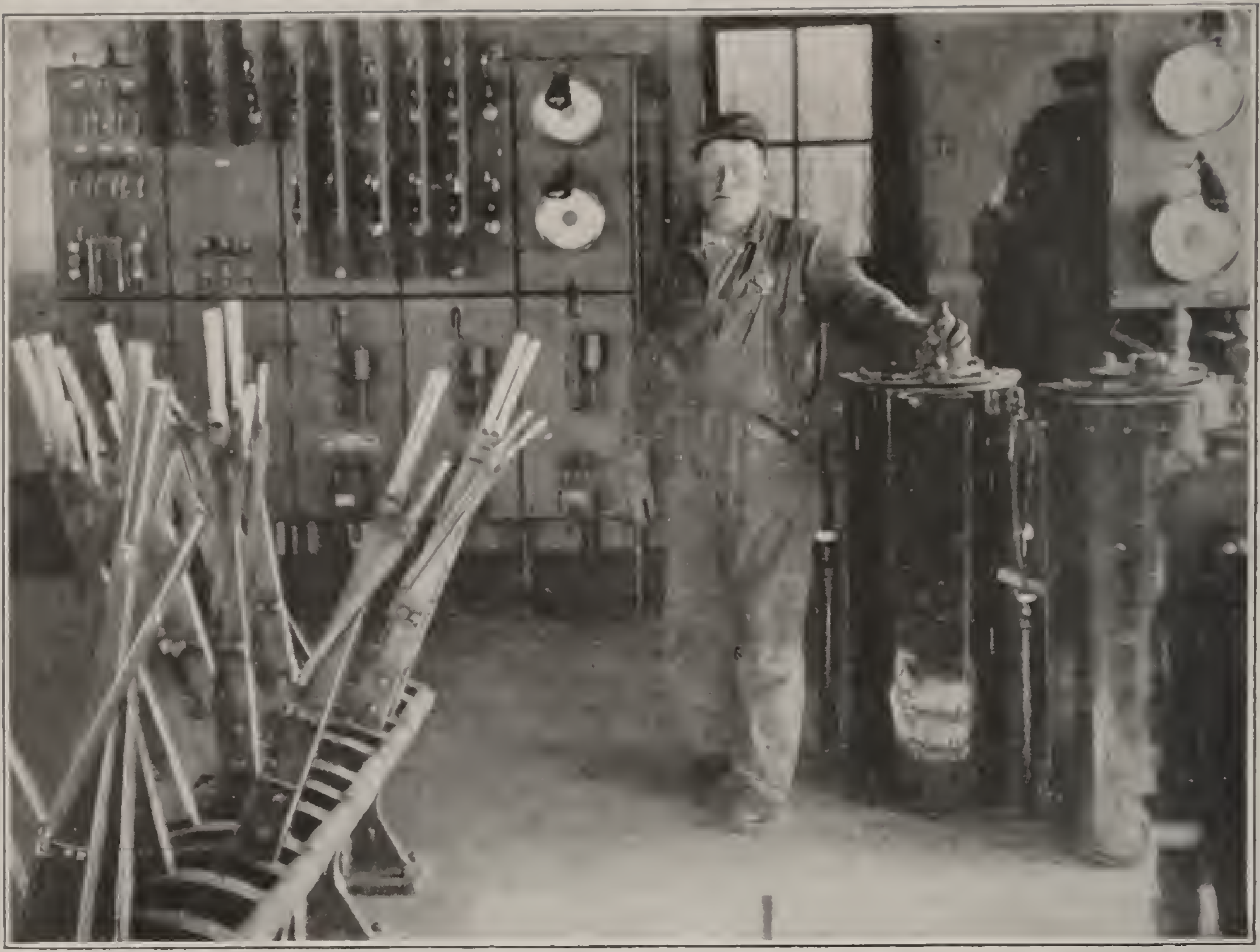


B. "BULL GEAR" READY FOR SHIPMENT.

C. INTERMEDIATE GEARS, HERRING-BONE TYPE.



MILLING THE TEETH OF A BULL WHEEL.



A. WINCH ROOM ON CONREY NO. 4 DREDGE.

B. WINCH ROOM AND LEVERS ON YUBA NO. 4 DREDGE.

the bull-wheel rims, one company cast them of chrome-nickel and the other company made them of manganese steel. Use of these steels not only increases greatly the resistance of the teeth to wear but insures against breakage. A 12-foot gear is as large as can be shipped by freight (Pl. XXIII, *B*) if cast in one piece, a consideration that no doubt accounts for retention of that diameter. The third company increased the diameter to 14 feet with a wider face and cut teeth, but cast the wheel in halves which could be conveniently shipped. The material used was carbon steel.

The pinions are made of the same material as the gears they drive. Each of these three designs will no doubt solve the problem of rapid wear, but only actual service will determine which is best.

The intermediate gears for heavy service should be made of chrome nickel and have cut teeth. One of the most recent dredges is using cut gears of the herring-bone type (Pl. XXIII, *C*), which gives a more even running gear and does away with much of the noise. With this type it is essential that the shafts be kept in perfect alignment and that provision be made against end thrust. On dredges with wooden hulls these requirements can not be met, but should be easily met on dredges with rigidly built steel hulls.

On the early small-size dredges the tumbler was driven by one bull-wheel gear, but the torsional strain set up in the tumbler shaft caused the shaft to break. At present practically all dredges have a gear on each end with two trains of gears driving through the intermediate shaft from the pulley shaft. The most difficult problem connected with the main drive is to get an equal pressure on each of these sets of gears. It is necessary to make arrangements so that adjustments can be made to meet the more rapid wear of some of the gears. On most of the dredges this adjustment is made possible by having the pulley shaft in two parts connected with a coupling. A device consisting of an equalizer or differential mounted on the pulley shaft now takes care of this adjustment automatically.

Another comparatively recent improvement is to have two short shafts instead of having the intermediate shaft take both gears. An advantage of this arrangement is that the hopper may be made larger where it goes between the shafts; formerly it was limited to a size such that it would not interfere with the intermediate shaft.

As mentioned in the chapter on electrical equipment, opinions differ as to advisability of driving by a direct-connected motor or by a belt, but obviously it is essential to provide a drive of some kind that will slip when necessary, and a belt seems the simplest means of accomplishing this end.

WINCHES.

Winches are used to raise the ladder, drive the bucket line, swing the dredge, raise the spuds and stacker, and for repair work. Different combinations are used, but the most general arrangement is to use a winch called the ladder-hoist winch for hoisting the ladder and driving the buckets. The swing winch contains the bow and stern swing drums, the spud hoist, the spare drums for wrecking and repairing, and also the head-line drum, if one is used. The stacker hoist is usually placed separately at the rear of the dredge on the upper deck, as this arrangement does away with a number of sheaves. Levers used in operating winches are shown in Plate XXV. Plate XXVI shows a dimensioned drawing of the starboard winch operating lever for a 9-foot dredge, and Plate XXVII shows the general arrangement of winch equipment on a 9-foot dredge.

LADDER-HOIST WINCH.

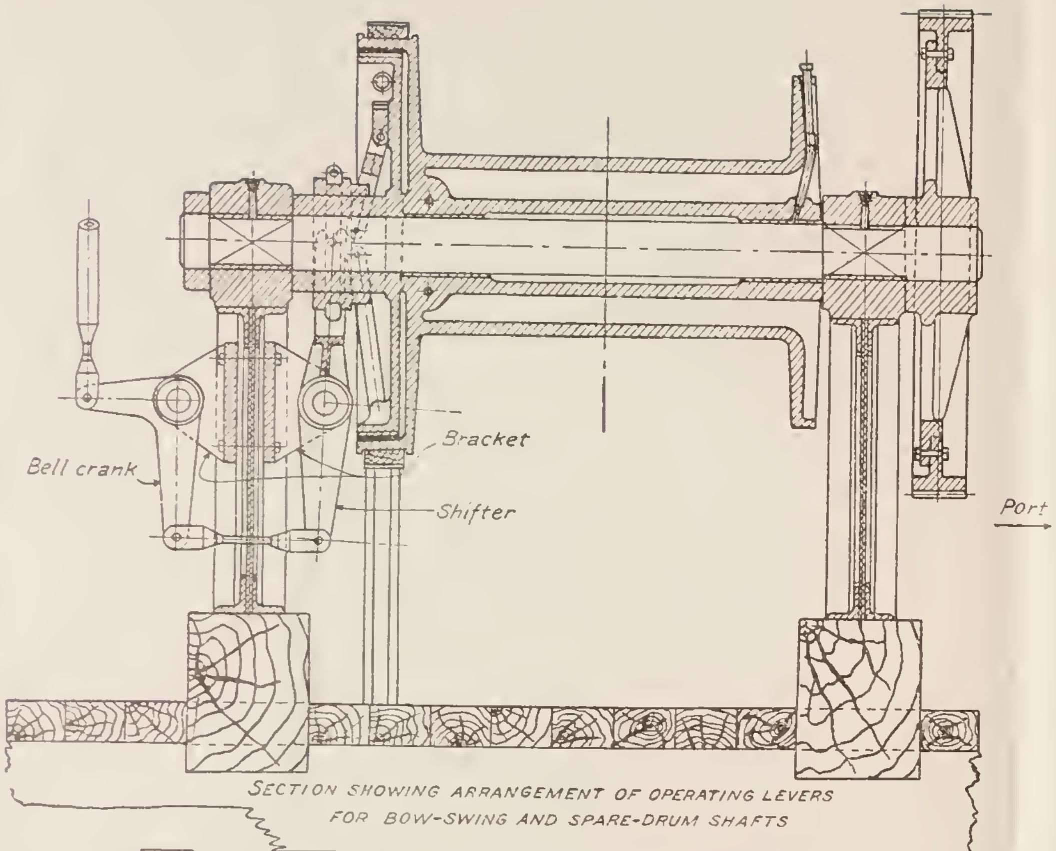
Some dredges have a separate winch for raising the ladder, the bucket-drive motor being placed at the main drive. Doubtless there are many advantages in having the winch for raising and lowering the ladder driven by its own motor and operated by a controller in the winch room, especially when the bedrock is uneven or where it is desirable to drop the ladder at certain parts of the cut, which is necessary in practically all dredging. If there is no separate winch for this it is necessary to raise and lower the ladder by means of levers, and the winch man may neglect to work the ground to the best advantage. The winch man should be able to operate the ladder and run the buckets independently of each other, and sometimes run the buckets while the ladder is being raised. Plate XXVIII shows a dimensioned drawing of the general arrangement of ladder-hoist winch equipment for a 9-foot dredge.

If the same motor is used to drive the bucket line and raise the ladder it must be large enough to perform both operations at the same time. When the bucket line only is being driven, the motor is not running economically, the load being much below its normal capacity.

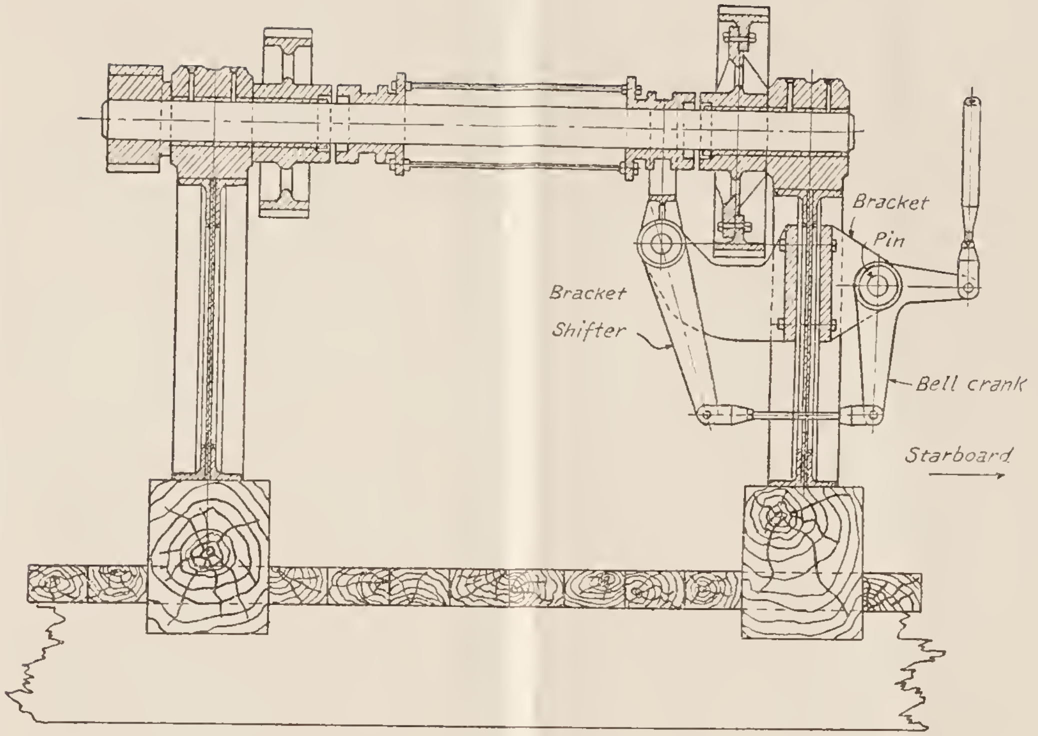
Another advantage of the separate hoisting winch (Pl. XXIX, *A*) is that there is no danger of dropping the ladder, a danger always present with the other type as the separate winch is so constructed that the motor must be run while lowering as well as while raising. The ladder is also lowered more evenly than with brakes.

When the ladder hoist and the bucket drive are combined, the brake bands should be placed directly on the drum; otherwise the chance of dropping the ladder is increased. The brakes should be powerful and they should be of the double-action type; that is, the tension from the hand lever should be thrown on the band on one side of the drum, which in turn should throw the increased tension on the other band.

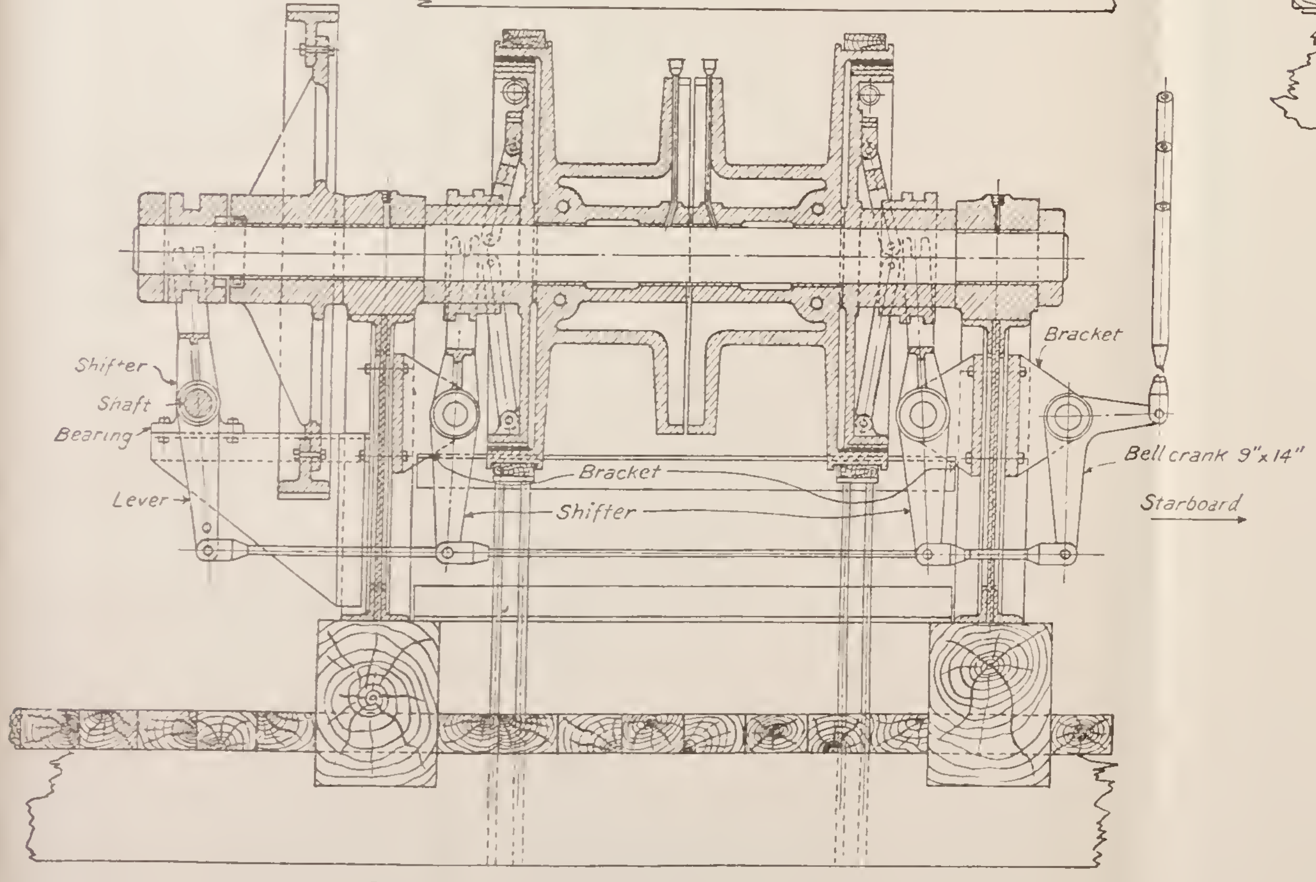
All revolving parts of the clutches should be built of cast steel. If made of cast iron and the ladder should drop faster than desired,



SECTION SHOWING ARRANGEMENT OF OPERATING LEVERS FOR BOW-SWING AND SPARE-DRUM SHAFTS

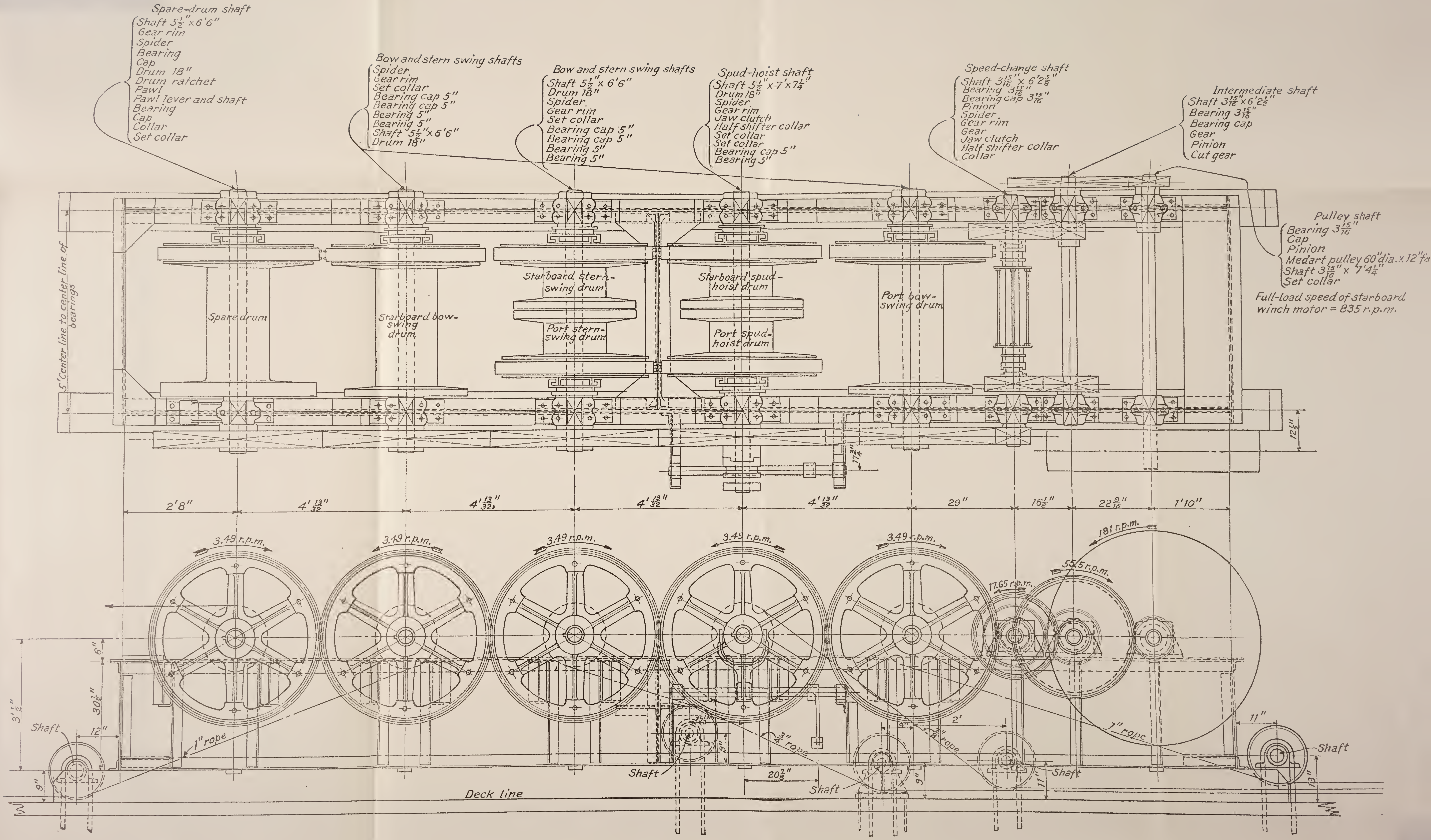


SECTION THROUGH SPEED-CHANGE SHAFT

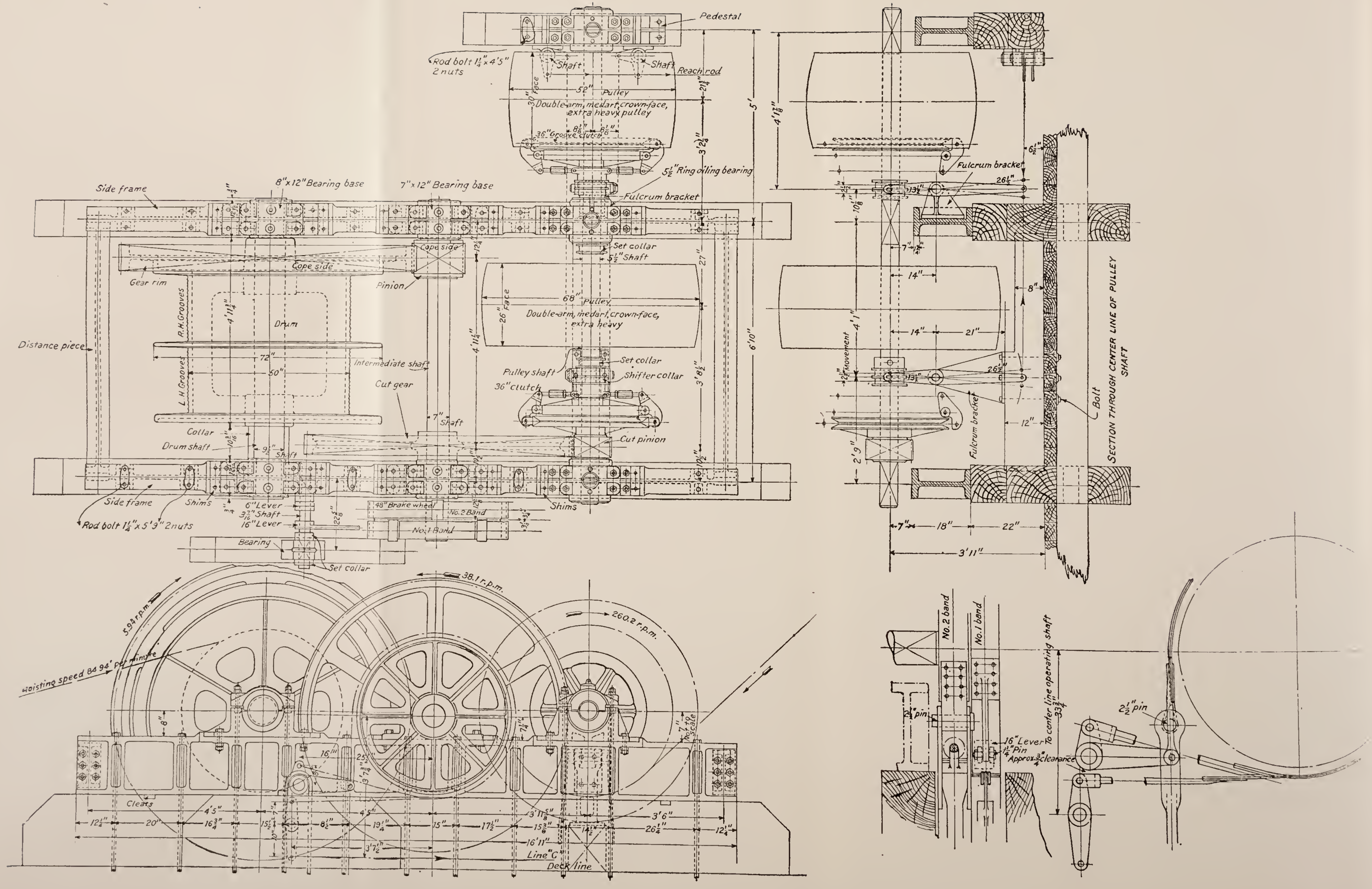


SECTION THROUGH SPUD-HOIST SHAFT

STARBOARD WINCH FOR 9-FOOT DREDGE.



GENERAL ARRANGEMENT OF WINCHES ON A 9-FOOT DREDGE.



LADDER-HOIST WINCH FOR 9-FOOT DREDGE.

the clutches would revolve at a terrific speed and fly into pieces owing to the centrifugal force. This accident has happened on a number of dredges and should be guarded against. The teeth of high-speed gears should be cut. The drums should not be grooved, as it is not practical to make the drum sufficiently large to take all of the hoisting rope in one layer. Therefore, as there are several wrappings of rope, the lead should be arranged so that the wrappings of the line will be close together with no space between; otherwise wedging of one coil in an open space may hold the rope briefly and cause the ladder to go down in short jumps, endangering the dredge.

Several serious accidents to the ladder have been caused by the breaking of the rods leading to the brakes. These rods are as a rule made of 2-inch pipe, the different lengths being fastened together with standard couplings. The pipes being thin, break at the threads, owing to the strain and the vibration of the dredge. The pipes should therefore be extra heavy and fastened together with special couplings.

Maple blocks are generally used in the friction clutches. These wear out in a comparatively short time, and some experiments have been made with asbestos blocks. The latter are made in an iron mold under great pressure. Small copper wires are run through the blocks to bind the asbestos together. The coefficient of friction of these blocks is higher than that of the wood blocks, they wear longer, and the first cost is not great, but the objection to them is they wear out the steel surface against which they rub.

SWING WINCH.

The swing winch (Pl. XXIX, *B*) should be of the eight-drum type with two drums on each shaft and a bearing at the center of each shaft as well as at each end. It has been the custom to have one drum at the front end of the winch for the spare drum, but it has been found that a great saving of time has been effected in repairing wrecks if two spare drums are available. If there are not two spare drums it is necessary to use one of the swing drums and to take the rope from that drum and to wind on another. The time saved in one use of the spare drum more than offsets the extra cost. Another advantage of having two drums for spares is that in case of trouble with one of the other drums it can be taken out for repairs and one of the spares substituted without a serious shutdown of the dredge. For this reason all shafts with the drums and clutches should be interchangeable. Some of the shafts are subjected to much heavier strains than others because of the duty they perform, and it is often the practice to make some of the winch shafts lighter than others, but this practice is not considered good.

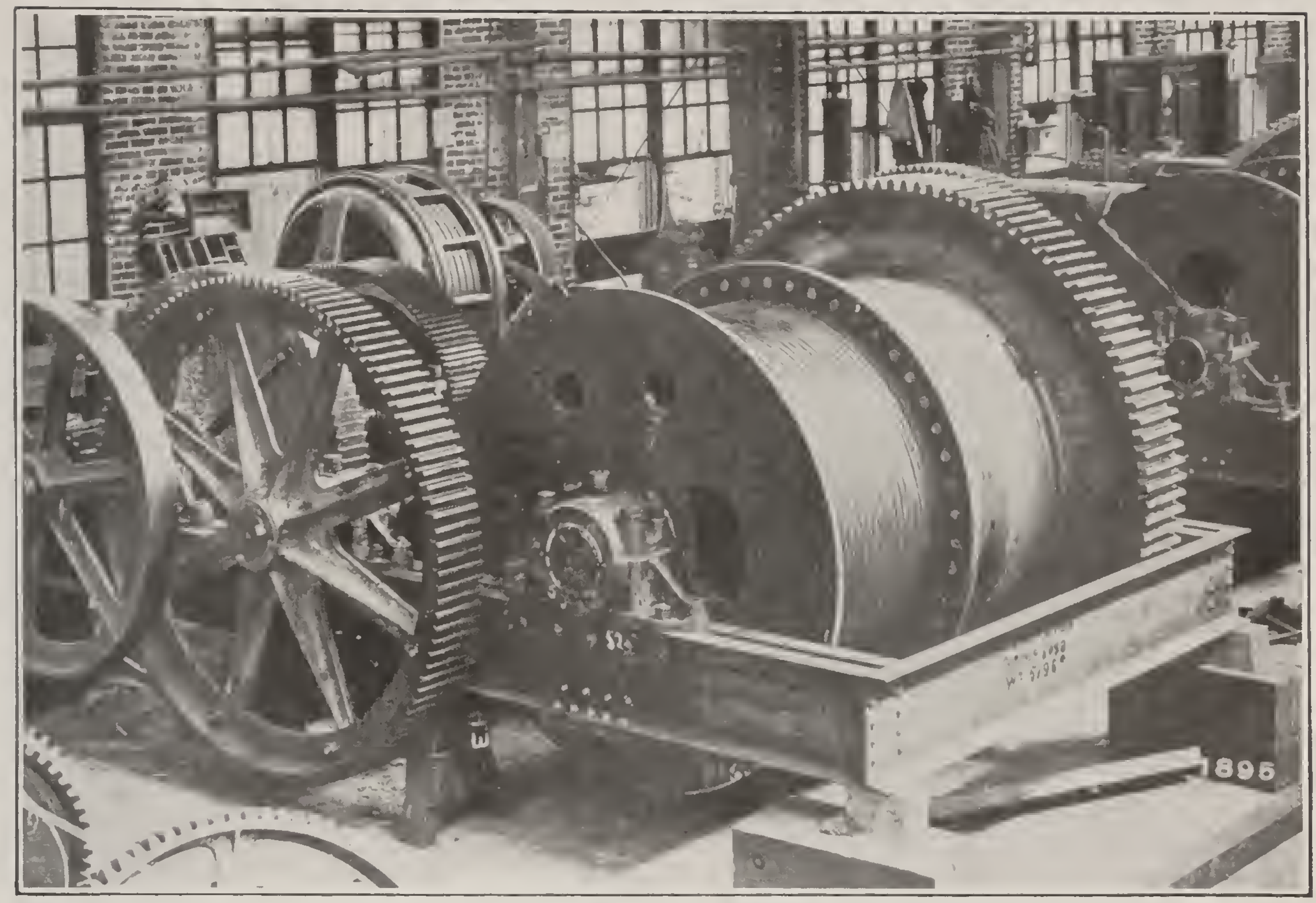
The spare drums should be placed at the bow end of the winch, the bow-swing lines next, then the spud hoist, and the stern-swing line drum at the stern end. The spud shaft should be arranged so that it will not revolve unless the spuds are being lowered or raised, as there is a constant tension on the drums causing the bushings to wear unnecessarily if the shaft revolves continuously. It is also desirable, but not so important, to arrange the spare drums in the same manner. The winch for a large dredge recently built was designed so that the bow drums and gear could be driven separately, the other gears remaining stationary except when desired. The arrangement proved satisfactory.

As the swing winch is usually placed on the starboard side of the boat, the port bow line must be run to the opposite side of the dredge. This is done either by leading the line to the stern and across the dredge amidships and then to the bow, or by bringing it forward and across the dredge through a sheave hung on the bow gantry. The latter arrangement is preferable, as the rope does not have to run around so many sheaves. If the first method is used, a very small sheave must be used to deflect the rope parallel to the deck, owing to the small amount of room under the winch; but when the rope is taken forward this deflecting sheave can be made as large as desired, as it is out in the clear. In taking the rope over the bow gantry a large sheave should be suspended at the center of the gantry. The size of the sheaves is of much more importance than is usually considered; it is discussed on another page under the subject of wire rope.

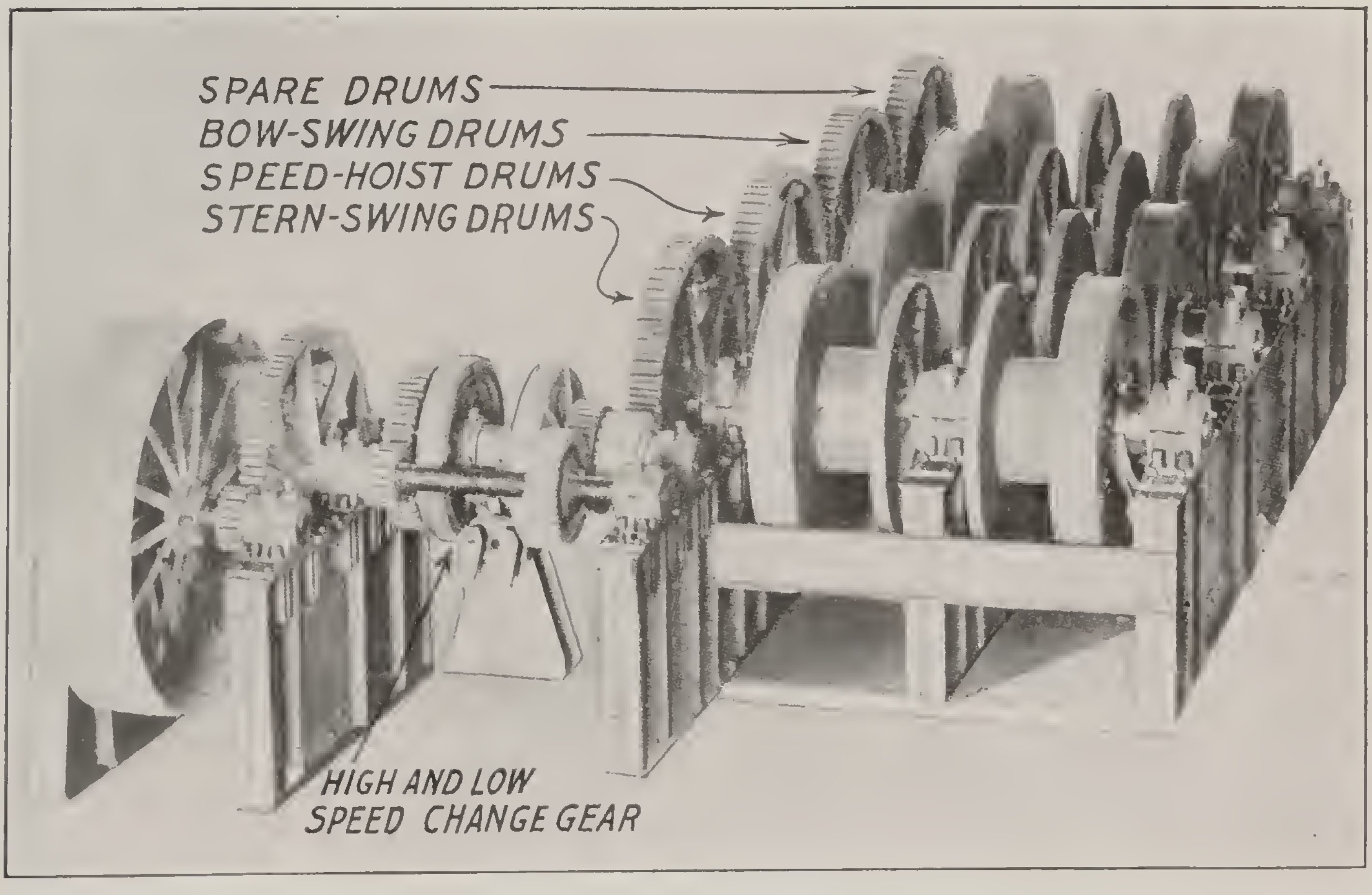
Most swing winches are constructed so that either of two speeds can be used for swinging the dredge. But in practice when the proper swinging speed has been determined the change speed is rarely used. In fact, most dredge operators disconnect it so that the winchmen will not be tempted to use it. For average ground the swinging speed at the lower tumbler is about 50 feet per minute. Experiments are being made by some of the larger companies with a view to increasing this speed, as it is realized that the buckets should be fed to the ground as fast as practicable. Strains in the side lines and other parts of the dredge are less when a rapid thin cut is taken than when a heavy slow cut is made.

When digging clayey ground it is especially desirable to take a thin cut as the clay is more easily broken in the screen when in thin slices than when in lumps.

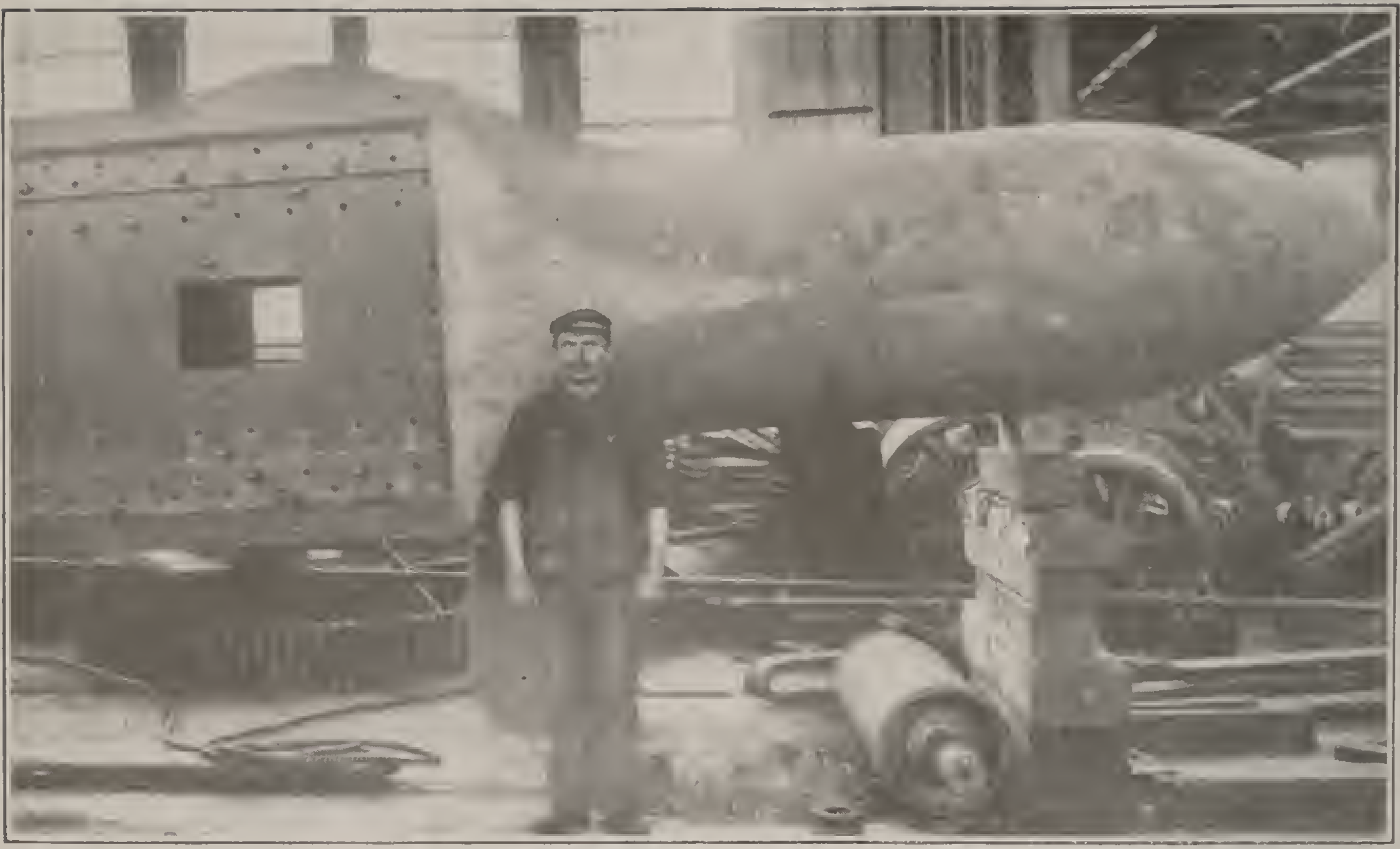
It is desirable to have a slow-speed mechanism on the winch so that in an emergency it can be used, for example, in repairing a broken ladder, or if it has been disconnected, it can be readily reconnected. On the older dredges this speed change was accomplished by jaw clutches in connection with two gear ratios. In modern large dredges such gear clutches have been displaced by friction clutches; these cost considerably more but are more easily operated.



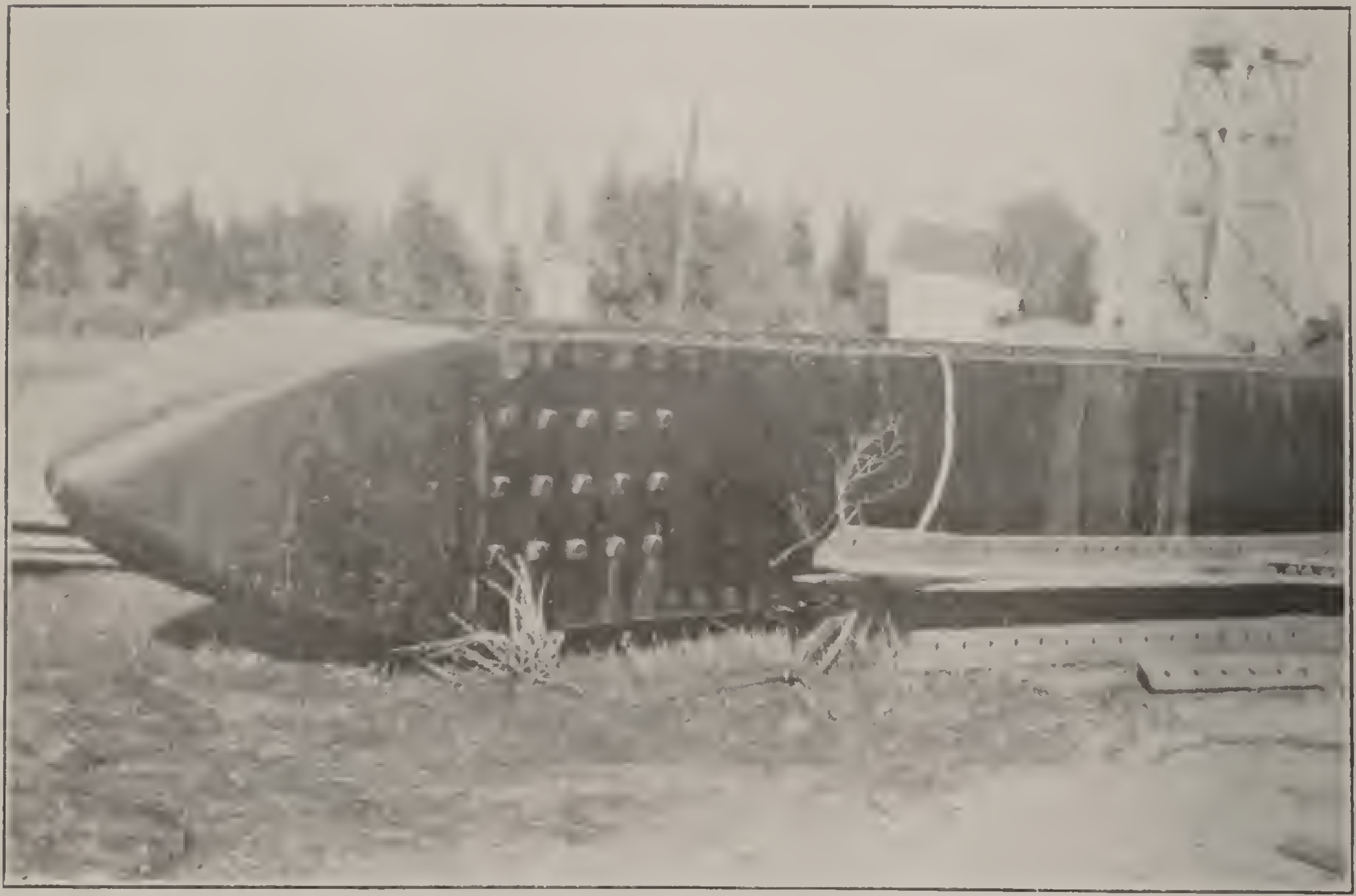
A. LADDER-HOIST WINCH FOR LARGE DREDGE, DIRECT CONNECTED.



B. STANDARD TYPE OF SWING WINCH.



A. CAST-STEEL ROUND POINT FOR SPUD.



B. STEEL SPUD, WITH SHORT SQUARE HEAD.

Most of the swing winches are driven with a belt, but the silent chain drive has been tried and will no doubt come into more general use. The winch motor should be supported above the winch, for when it is set on the deck there is always danger of water getting into it when the decks are washed.

On some dredges the designers have saved on the first cost by using jaw clutches instead of friction clutches on each drum, but these have not proved satisfactory, as time is lost at each swing of the dredge in getting the jaws in mesh. Another disadvantage is that the dredge is reversed with a jar. This jar is reduced by using a friction clutch. As clutches on dredges are subjected to severe service, the standard type of friction clutch is not strong enough, so a special type of internal expanding band clutch has been developed for this work and has proven efficient.

SPUDS AND SPUD SUPPORTS.

The most important item to consider in connection with spuds is strength. Spuds not only take the thrust caused by digging but they are also subjected to a torsional stress from the swinging of the dredge from side to side. The torsion can be reduced to a minimum by making the spud-point casting very long, and the cross section round except where the point is riveted to the structural-steel part of the spud. The spud is in the sand to a depth of several feet, and the round point (Pl. XXX, *A*) offers less resistance to the swinging of the dredge than does the rectangular section that was until recently in general use.

The designing of a sufficiently strong spud would seem simple, the spud being in effect an overhanging beam supported at the hull and the upper truss chords. However, with perhaps one or two exceptions, a spud has rarely stood up during the life of the dredge for which it was built.

Wooden spuds were used for the early dredges, but were not found suitable. Later, some dredges used a steel spud for digging and a wooden spud for stepping ahead, and some dredges even now use that arrangement. This practice is not good for two reasons—first, either spud should be available for digging as conditions often arise that make the use of both necessary, and second, there should always be a spare spud for use in digging if breakage occurs; the spare should be carried on the dredge, as often it can not be transported to the dredge. Thus, if one spud cracks or the rivets loosen, digging on it can be discontinued before it actually breaks and it can be used as the stepping spud until the dredge is in a suitable location to permit its removal for repairs or the substitution of a new spud.

There are two types of steel spuds in general use, one having a rectangular cross section (Pl. XXX, *B*, and fig. 6) and the other of the same general shape, but with a center web running the length of the spud (fig. 7). The latter type is much more difficult to build and to repair. Tests have been shown that if the center web be omitted and the metal used to thicken the outside plates, the life of the spud is much longer. On a large dredge recently built an experimental spud is made of four 30-inch I beams bolted together (fig. 8).

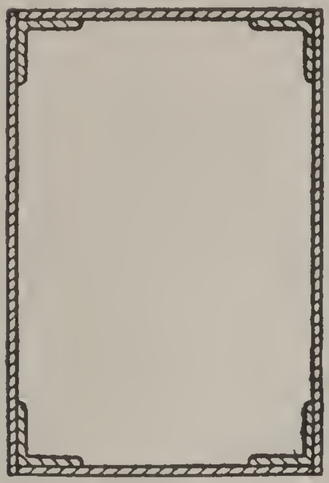


FIGURE 6.—Cross section of box type of spud.

Breakage of many spuds theoretically strong enough is caused by the rivets becoming loose and allowing the plates to fail. It is difficult and often impossible to keep the rivets tight on most of the dredges. The tremendous bearing pressure on the spud kick is distributed over three or four rivets only, causing them to become loose. During the past year most of the large dredges have been designed to overcome this by making a swivel casting which bears against the spud at all positions for a distance of 3 or 4 feet, thereby distributing the bearing pressure over a great number of rivets. Plate XXXI, *A*, shows one method of transporting a spud.

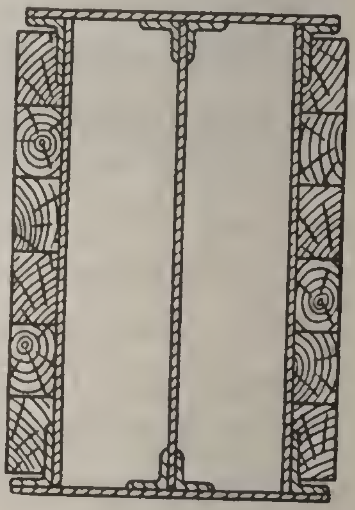


FIGURE 7.—Cross section of spud with center web.

The life of most dredge hulls is shortened by their failure at points where the spuds bear against them. In many dredges there is a steel plate between the hull and the spud. On such dredges the plate is apt to be driven into the wooden hull, the intermittent pressure loosening the bolts in the hull and allowing the water to leak through. In some of the steel dredges trial was made of a small casting riveted to the hull plates, but the result was the same. It is now the practice to rivet a casting several feet wide the entire depth of the hull.

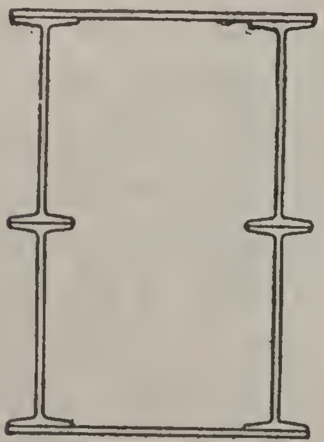
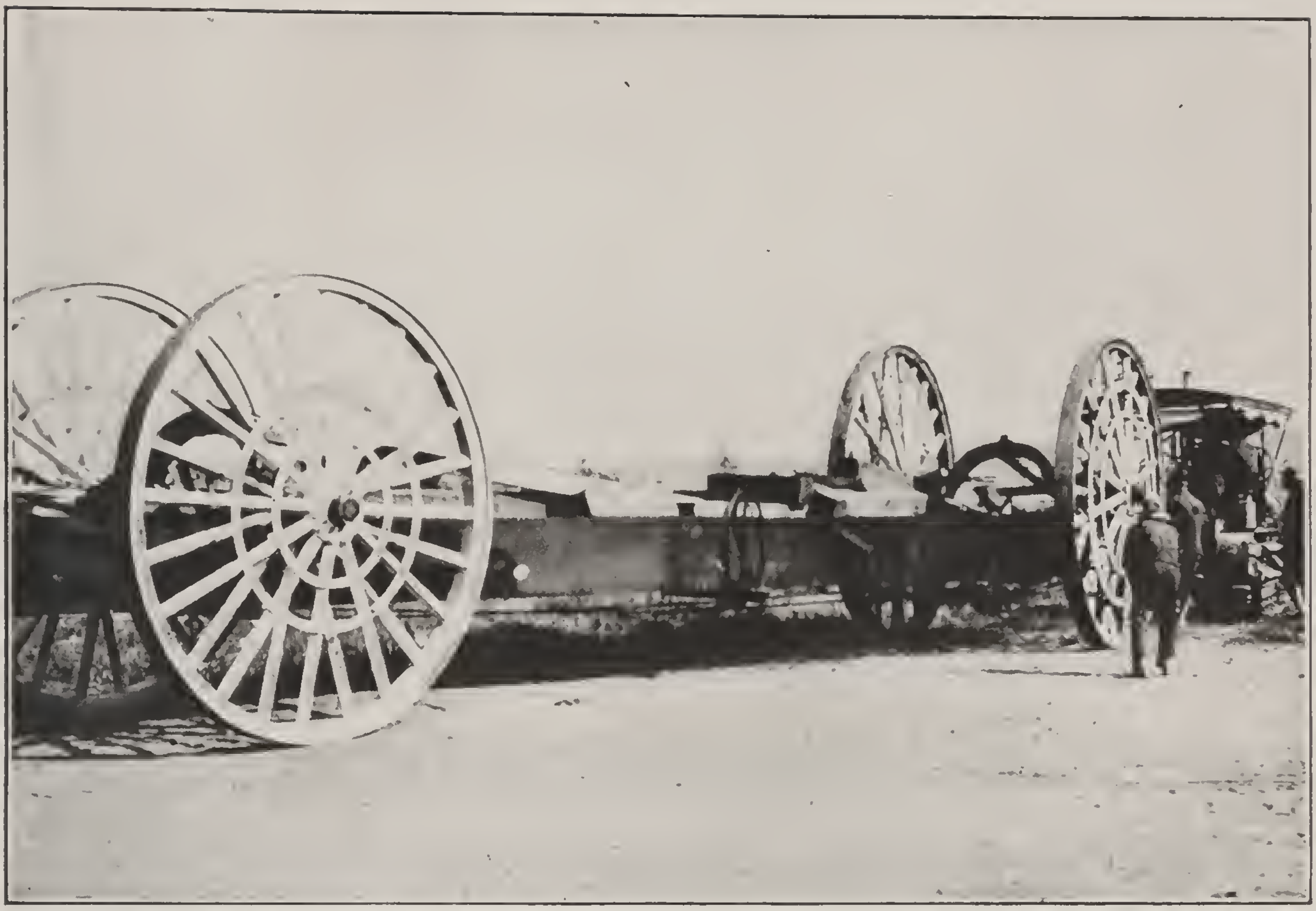
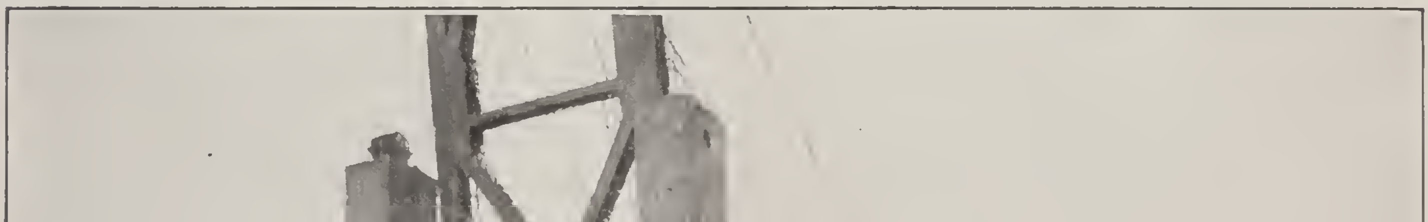


FIGURE 8.—Cross section of spud made of four 30-inch I beams bolted together.

For a while wooden casings just large enough to receive the spud were built on the hull from the upper truss to the deck. Timbers were bolted to the spuds on the sides. The casings served no good purpose and caused trouble. Small stones, nuts, or other pieces of material dropped between the casing and the spud and necessitated the use of more power to raise the latter. Casings should be left open except at the upper truss chord and at the hull where the stresses can be taken care of. It is the usual practice to use buffer springs at the upper supports to do away with excessive jar (Pl. XXXI, *B*). Springs have also



A. HAULING A SPUD.



B. EQUALIZING SUPPORT FOR SPUDS.

been used at the hull, but as it is necessary to put them under the water, where it is impossible to examine them at intervals, the practice has been discontinued, although they never gave trouble.

Several spuds have been bolted together instead of riveted, lock washers and sometimes turned bolts which were made a driving fit being used. Bolts stay tight much better than rivets and if they become loose they can be tightened. They serve the purpose very well and will no doubt do away with much of the trouble experienced with rivets.

A single digging spud was placed on each of several dredges with the idea that by placing it in line with the bucket ladder, to oppose the digging force, the stresses in the dredge could be more effectively distributed. It is possible to take care of the eccentric load caused by the spuds being placed off center in the usual manner, so there seems no good reason why the single spud should be used. With these dredges a light stepping spud was placed as in the standard dredge. The use of only one digging spud entails many disadvantages, and the dredges so equipped have given much trouble—the spud comes at the end of the screen and all of the material has to be by-passed through a curved chute; the stacker must be placed to one side and run out at an angle, which complicates the suspension, and the spud is in the way when the screen and the drive are being repaired. A still greater disadvantage is that when the spud is out of order or the hull at the spud needs repairing, the dredge is shut down; this is not necessary if the dredge is equipped with two spuds. When two spuds are used and the rivets on one become loose, the other is put into use and the loose rivets are tightened while the dredge remains in operation.

The length of the spud is determined by the depth of digging and the height of the top truss chord. The following table shows how the sizes of spuds vary according to the ideas of the constructors and the conditions of operation:

Data regarding lengths of spuds used on different dredges.

Size of bucket.	Size of spud.	Depth of digging below water.	Length of spud.
Cu. feet.	*Inches.*	*Feet.*	*Feet.*
4	26 by 36	35	59
5	24 by 36	50	53
5	24 by 36	30	45
6	24 by 36	60	53
7	24 by 36	30	48
9	24 by 36	65	61
13	34 by 54	45	60
15	30 by 42	35	61
16	38 by 50	65	70
16	36 by 54	55	80
16	38 by 60	70	62
16	36 by 54	40	60

HULLS AND SUPERSTRUCTURES.

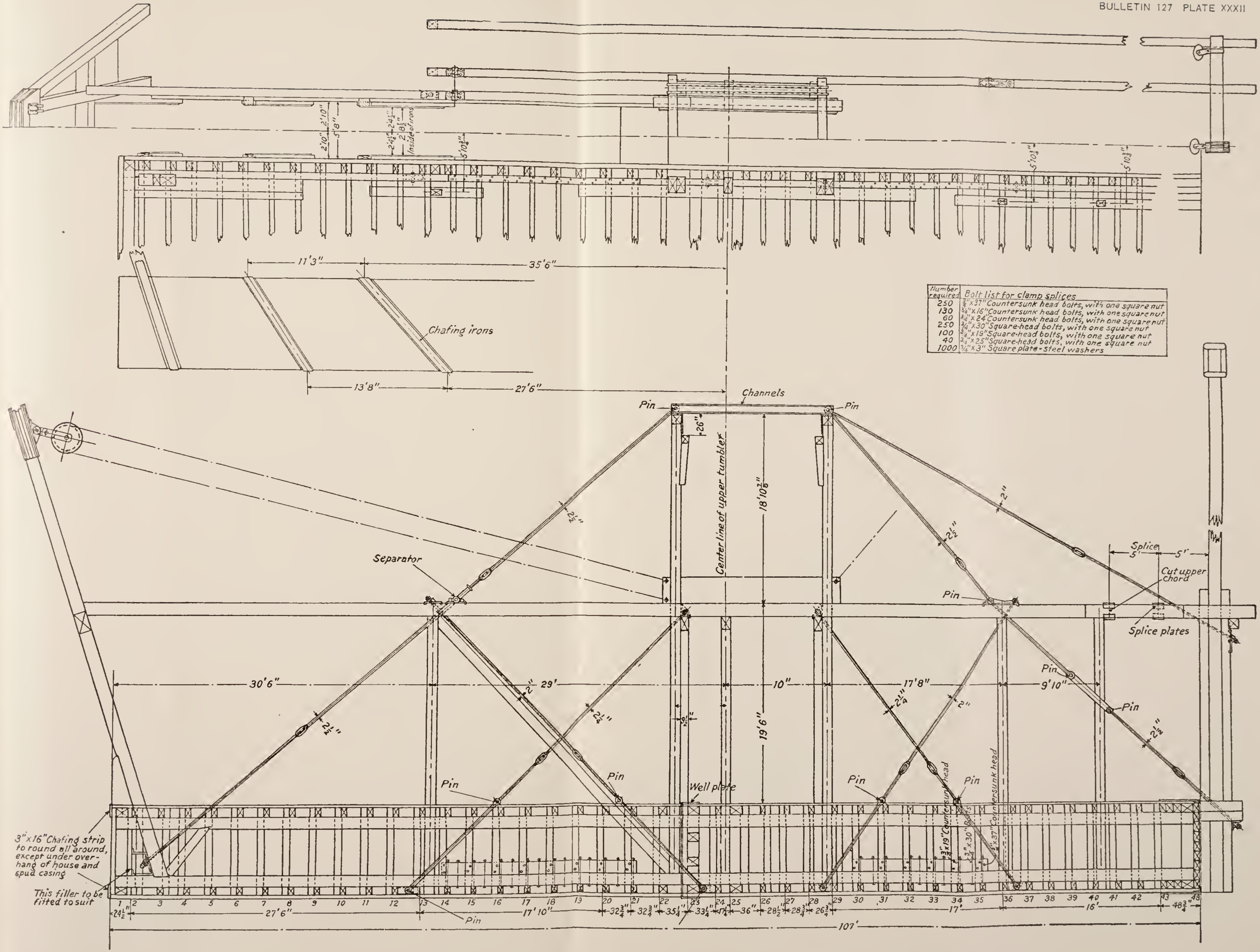
The hull is rectangular, with an opening or well hole running from the bow to about the center. The ladder and bucket line extend through the well hole, and the well hole should be no longer than is absolutely necessary for the operation of the buckets when the ladder is in a position 45° with the vertical, this being the maximum angle at which it should operate. The outside corners of the pontoons are cut off at the bow on a long curve. The object of making the hull narrow at the bow is to avoid interference with the bank when digging in corners. The early California dredges dug shallow ground; the ladder did not extend much ahead of the hull, and it was necessary to make the hull narrow at the bow. With long digging ladders the corners should be cut as little as possible, and under certain conditions they need not be cut off at all. The most important reason for this is that a larger proportion of the ladder is supported by the bow gantry which in turn is supported by the forward end of the hull, and it is necessary to get as much flotation at this point as possible. Consequently, the front wall of the hull should be square with the deck and not beveled, as on some dredges. Plate XXXII shows an elevation of a dredge hull and Plate XXXIII shows details of hull construction.

If much sand is to be encountered at the stern of the dredge, it is good practice to cut off the corners at the bottom of the stern, so that when the dredge is swung these corners will be clear of the sand tailings (see Pl. XXXIV, *A*). The deck is usually allowed to extend over the hull 3 to 5 feet on each side, but this extension is desirable only if there is not sufficient room on the deck for the winches and pumps.

Hatchways are placed in the deck to give entrance to the hull at different points, but they should be arranged so they are water-tight when the dredge is in operation, so that if the hull is injured it will not fill with water.

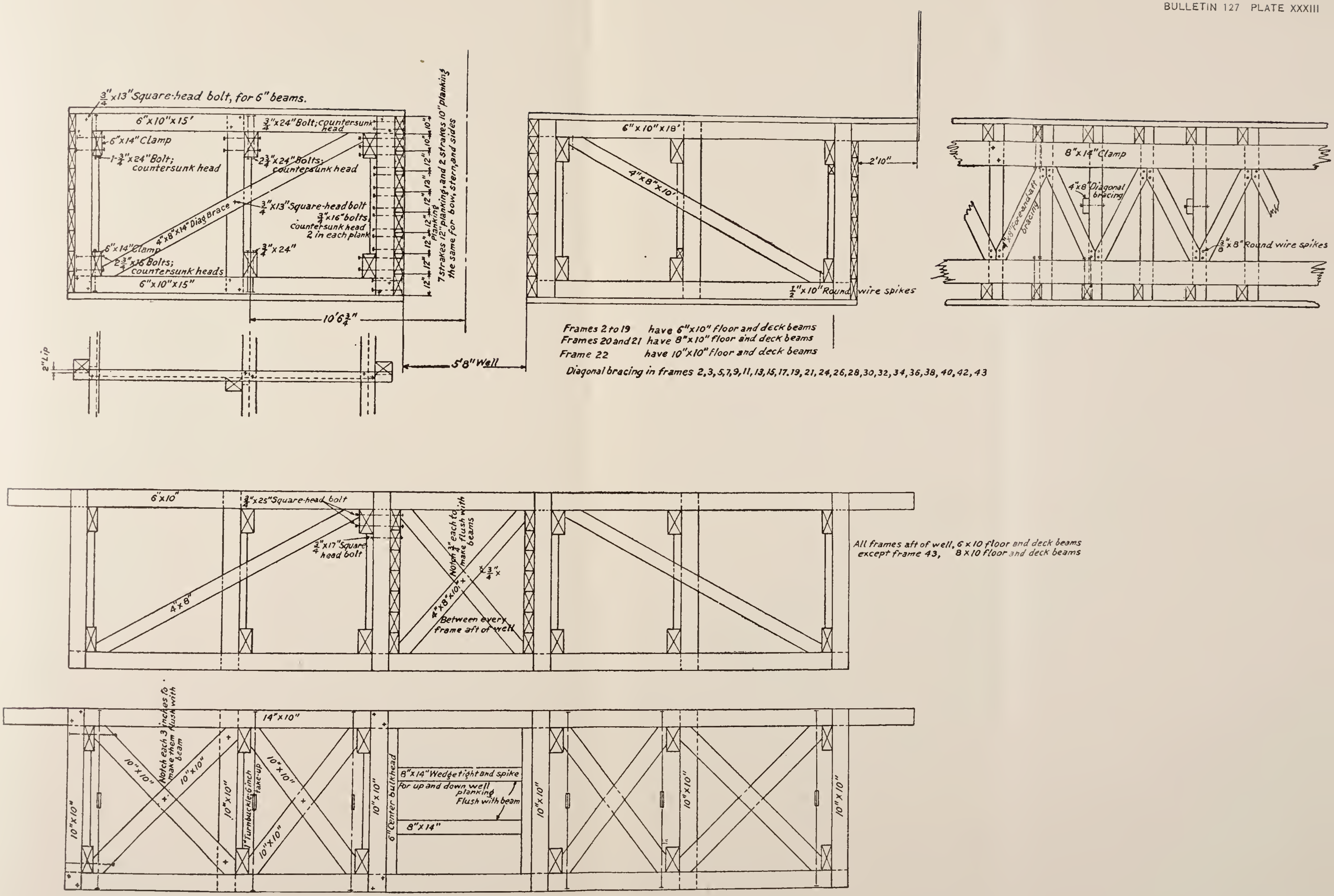
The material of which the hull should be built is determined by local conditions, such as the comparative cost of steel, whether suitable timber is near the site of operation, the life of the dredging ground, the availability of skilled labor, and whether parasites or chemicals are present in the water. The greatest advantage obtained by the use of steel hulls (Pls. XXXIV, *B*, XXXV, and XXXVI) is the elimination of the fire hazard, thus not only forestalling losses occasioned by fire but effecting a great saving on insurance. The steel hull has other important advantages.

The life of the property is, however, not such an important item as is sometimes considered, for there is no reason why a wooden hull should not be built to last 15 years, and by that time the machinery



Number required	Bolt list for clamp splices
250	$\frac{3}{4}$"x37" Countersunk head bolts, with one square nut
130	$\frac{3}{4}$"x16" Countersunk head bolts, with one square nut
60	$\frac{3}{4}$"x24" Countersunk head bolts, with one square nut
250	$\frac{3}{4}$"x30" Square-head bolts, with one square nut
100	$\frac{3}{4}$"x19" Square-head bolts, with one square nut
40	$\frac{3}{4}$"x25" Square-head bolts, with one square nut
1000	$\frac{3}{4}$"x3" Square plate-steel washers

ELEVATION OF WOODEN HULL OF A 6-FOOT DREDGE.


$\frac{3}{4}$"x13" Square-head bolt, for 6" beams.
6"x10"x15'
6"x14" Clamp
1-$\frac{3}{4}$"x24" Bolt; countersunk head
$\frac{3}{4}$"x24" Bolt; countersunk head
2-$\frac{3}{4}$"x24" Bolts; countersunk head
4"x8"x14" Diag Brace
$\frac{3}{4}$"x13" Square-head bolt
$\frac{3}{4}$"x16" bolts, countersunk head 2 in each plank
6"x14" Clamp
2-$\frac{3}{4}$"x16 Bolts; countersunk heads
$\frac{3}{4}$"x24"
6"x10"x15"
Planking
10"
10"
12"
13"
12"
12"
12"
12"
12"
7 strakes 12" planking, and 2 strakes 10" planking the same for bow, stern, and sides
10'6$\frac{3}{4}$"
2" Lip
5'8" Well
6"x10"x18'
4"x8"x10'
2'10"
$\frac{1}{2}$"x10" Round wire spikes
8"x14" Clamp
4"x8" Diagonal bracing
4"x8" Fore and aft bracing
$\frac{3}{8}$"x8" Round wire spikes
6"x10"
$\frac{3}{4}$"x25" Square-head bolt
$\frac{3}{4}$"x17" Square head bolt
4"x8"
Notch $\frac{3}{4}$" each to make flush with beams
4"x8"x10'
$\frac{3}{4}$"
Between every frame aft of well
14"x10"
10"x10"
Notch each 3 inches to make them flush with beam
10"x10"
10"x10"
10"x10"
1" Turnbuckle 6 inch take-up
10"x10"
10"x10"
10"x10"
6" Center bulkhead
8"x14" Wedge tight and spike
For up and down well planking
Flush with beam
8"x14"
10"x10"
10"x10"
10"x10"

Frames 2 to 19 have 6"x10" floor and deck beams
Frames 20 and 21 have 8"x10" floor and deck beams
Frame 22 have 10"x10" floor and deck beams

Diagonal bracing in frames 2, 3, 5, 7, 9, 11, 13, 15, 17, 19, 21, 24, 26, 28, 30, 32, 34, 36, 38, 40, 42, 43

All frames aft of well, 6 x 10 floor and deck beams
except frame 43, 8 x 10 floor and deck beams

Frame 23
Frame 25 Same, except to have two 4"x8" diagonal bracing between bulkheads

HULL TIMBERS OF A 6-FOOT DREDGE.

will have outlived its usefulness, not only on account of wear but because it will have become obsolete.

The usual life of wooden hulls is only 8 to 10 years, as practically all of them are built of wood that has not been treated in any way by preservatives or has had only an external coating of preserving liquid. Careful tests have been made by the Government with rock scows and barges which operate under conditions similar to those encountered by dredges, and it has been determined that when built of untreated timbers they will last approximately eight years. However, barges built of timbers whose pores had been thoroughly filled with creosote by the vacuum and pressure process were excellently preserved at the end of 15 years. There is no doubt that all wood hulls should be built of creosoted timbers regardless of the probable life of the dredging operations. The additional cost for timbers treated in this manner is about 5 per cent. The timbers used in construction should also be as dry as possible; some constructors contend that dredge hulls built in the winter season will not have as long life as those built in dry weather.

It is often necessary to replace timbers that have been subjected to heavy strains or that have rotted from moisture. On some dredges the upper truss chord is under tension and bending strains due to the weight of the main drive. As timbers should never be depended upon to take tension, rods should be placed to take this strain. Timbers should not be used to support the screen, as the leakage causes rapid decay, and the vibration and the weight force the bearings out of line. It is not uncommon to replace the screen timbers every three or four years. These supports should therefore be built of structural steel, which is more economical and more satisfactory in every way than wood.

The truss chord and screen supports are the only timbers that give trouble above the deck, as the bow and stern gantries are usually in good condition when the dredge is dismantled except at the deck and where they project into the hull. These timbers as well as the truss posts should have a casing built around them at the deck, into which asphaltum or other material can be poured to keep out moisture. All heavy timber should be set in cast-steel or cast-iron bases at the bottom and have caps at the top, not only in order to keep out moisture, but to prevent the end of the timber from crushing the supporting member. Each of these bases and caps should have about three times as much bearing surface along the cross timber as on the end of the uprights; Douglas fir or Oregon pine, for example, should not have to support a weight of more than 250 pounds per square inch across the grain, an endwise pressure of 800 pounds per square inch is safe.

Decay always sets in at contact surfaces when moisture is present and extreme care must be used in protecting such places. Probably, however, the greatest source of trouble is the practice of running the truss rods through the timbers, as the water from the deck runs down along these rods into the timbers. Double truss rods should be used and should be placed outside of the timbers and connected at the ends to the bases and caps. This arrangement is somewhat complicated but it should always be followed.

The life of practically all wooden hulls is determined by the life of the stern, where the spud thrust is taken up. As mentioned under the subject of spuds, the latter should not bear directly against the hull, but a casting should be placed between in order to distribute the bearing strain to all of the main timbers running lengthwise of the dredge.

A mechanical circulation of air through the hull considerably prolongs the life of the timbers. The hull should be so designed and the machinery so placed that the bow of the dredge is slightly higher than the stern when the ladder is in the highest position, so that at all times the water will drain to the stern, where it can be pumped out.

Data on sizes of hulls for dredges of different capacities and digging depths are given in the table following:

Data on hulls of different dredges.

Size of buckets.	Digging depth.	Material of hull.	Width of hull.	Length of hull.	Depth of hull.
Cubic feet.	*Feet.*		*Feet.*	*Feet.*	*Ft. in.*
5	35	Wood...	36	106	7 9
5	55	...do....	36	115	8 0
7	25	Steel....	40	115	8 0
7	65	....	42⅔	130	8 6
9	25	Wood...	46	130	8 6
9	65	...do....	50	145	9 0
13	35	...do....	56	135	[a]14 6 [b]12 0
16	65	...do....	58	155	12 6
16	65	Steel....	58	155	12 0

[a] Bow. [b] Stern.

A hull constructed of steel can be divided into several water-tight compartments, so that if one compartment suffers accident the water is confined to it. The most vulnerable parts are at the bow and stern, and therefore a compartment is built at the bow on each side, another compartment is built across the stern, and others are constructed on each side and at the end of the well hole.

BOW GANTRY.

The bow gantry should not be placed near the front edge of the hull, but back a considerable distance so as to give as much flotation surface as possible under the gantry, as mentioned previously.

The grizzlies are usually protected by heavy cast-steel rails set about 18 inches apart, so that if the bucket line should break and the broken end go over the upper tumbler the grizzlies would not be damaged.

When it is necessary to dig the gravel to a depth of 50 feet or more a bucket idler (Pl. XL) should be placed to take a part of the weight of the return string of buckets. The idler is placed on the lower deck high enough so that the lips of a bucket will just clear the grizzly when the bucket is riding on the idler. This arrangement also affords a protection for the grizzly. A bucket idler is about 9 feet in diameter and the tread is shaped to conform to the edge of the lips of the buckets. The tread is shod with manganese-steel wearing plates. Idlers were formerly cast in four parts. The two spiders were cast separately and the rim was cast in two parts which were later bolted together. The shoulder of the rim against the ends of the spiders was also held with large bolts. The constant hammer of the buckets wore away the shoulder, when the bolts would not hold tight. At present the idler is cast in one piece, doing away with the difficulty mentioned.

The hubs of the idlers are long—in some idlers being more than 7 feet. Consequently it has been found difficult to keep the keyways from wearing loose, as the long hubs make difficult the use of taper keys. It is believed that this trouble will be overcome by splitting the hub the entire length and wedging it apart while boring it, and after the shaft has been put in place the wedges can be removed and the hub will bind on the key. The idler should not be pressed on the shaft. When repair of the idler is necessary it must be taken off the dredge through the well hole, and with the shaft in place it is too wide to go through the well hole. Hence the shaft must first be removed before the idler can be taken off the dredge. Similarly, in designing the idler, care must be taken to make it narrower than the well hole. In removing the idler it is lashed to the lower run of the buckets and the bucket line is revolved until the idler is landed on the bank.

SCREEN AND SCREEN DRIVE.

As the shaking screen is no longer used except in isolated dredging, only the revolving screen (Pl. XLI) is discussed here.

The size of the screen to be used depends upon the capacity of the buckets, the proportion of fine material, and the size of material that goes over the tables. The following table shows the sizes of screens with $\frac{3}{8}$-inch or $\frac{1}{2}$-inch perforations most commonly used for buckets of different capacities.

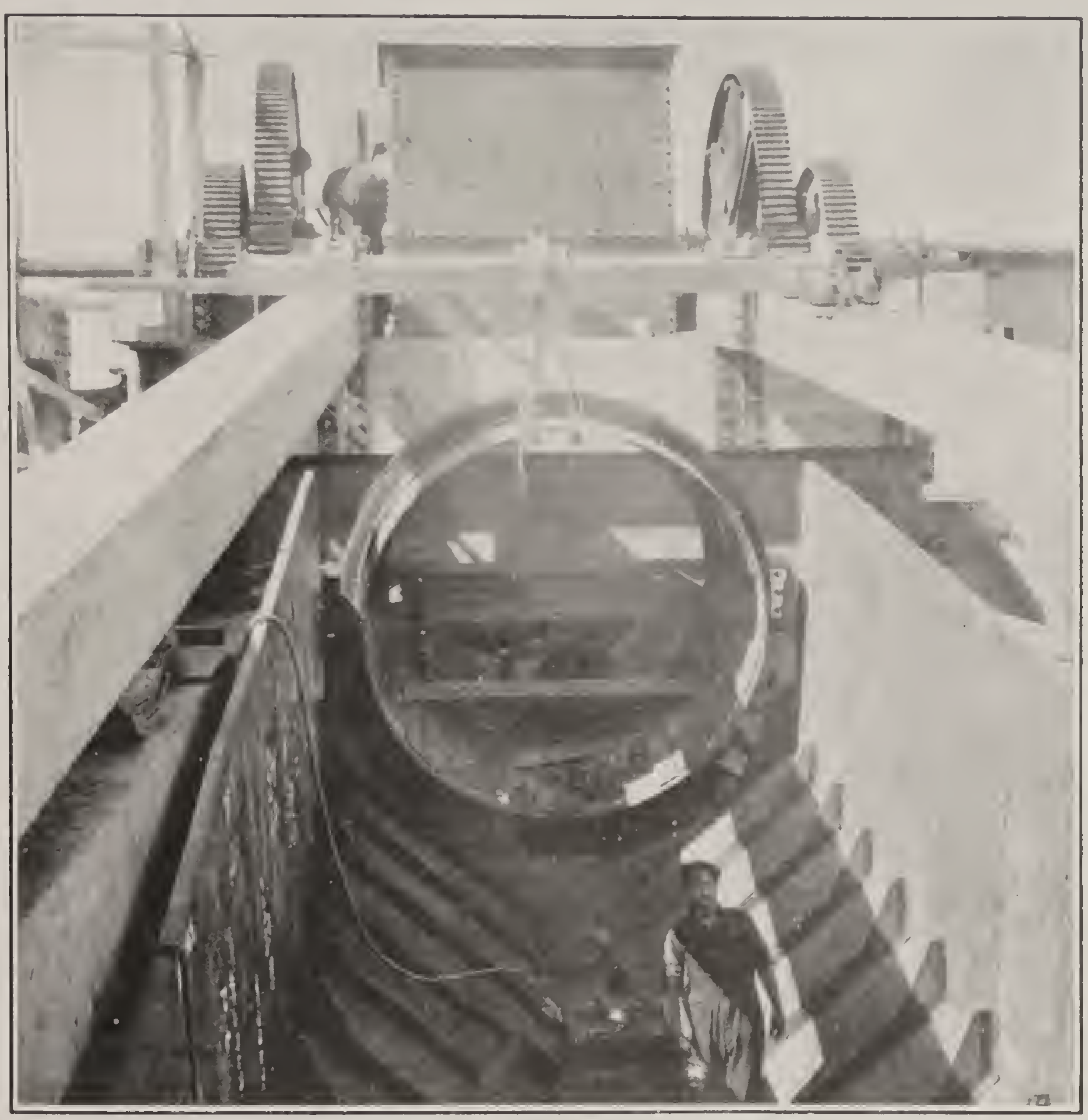
Sizes of screens used with different sizes of buckets.

Size of bucket.	Diameter of screen.	Total length of perforated plates.	Horsepower of drive motor.
Cubic feet.	*Feet.*	*Feet.*	
3	4	15	20
5	5	20	25
7	6	25	35
9	7	30	45
16	9	38	75

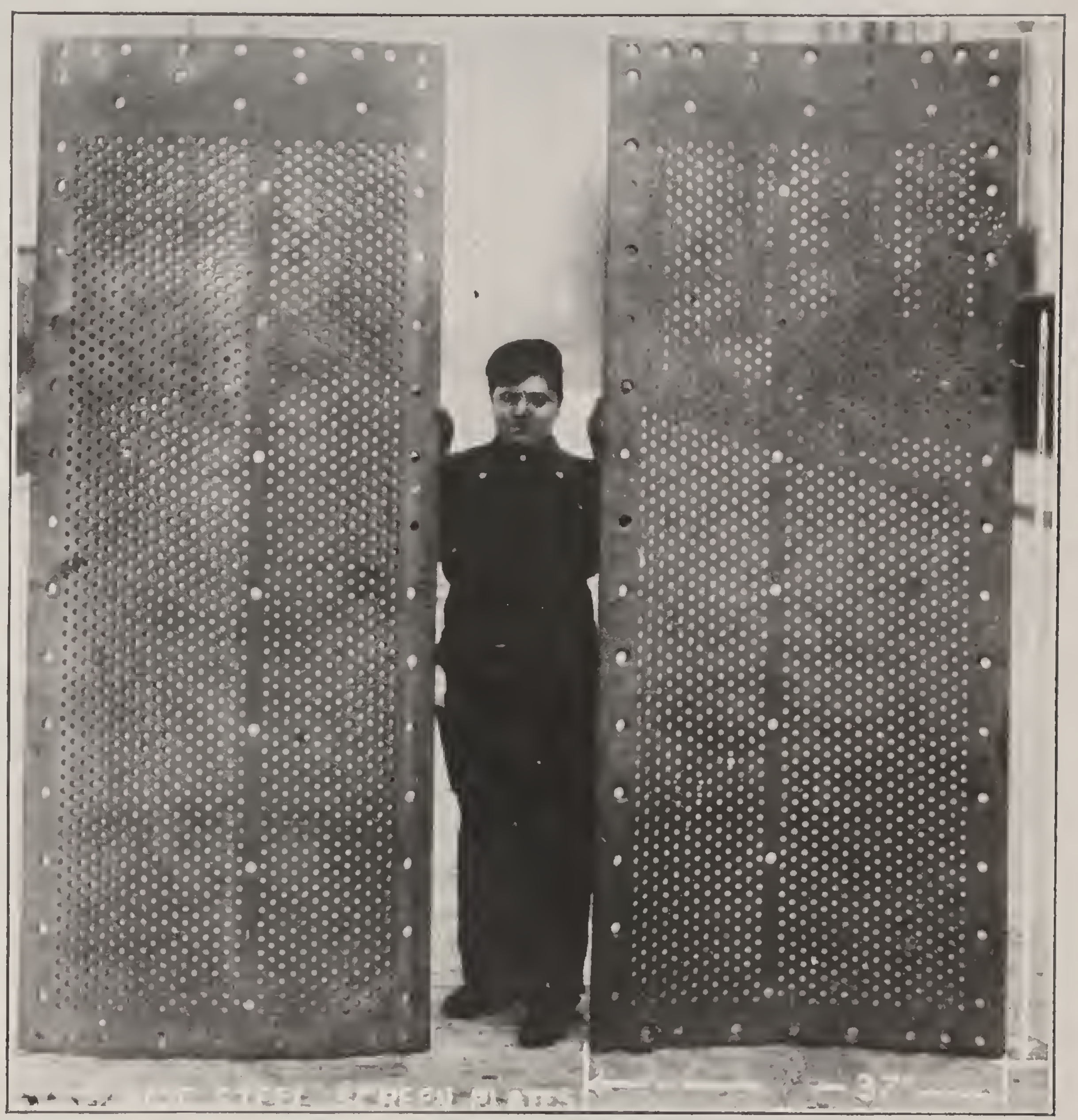
Screens are usually cylindrical, although some step screens are used. Each type has its advantages. Some of the advantages of the step screen are: (1) Weight is about 20 per cent less for equal capacity; (2) with the drive at the head end the weight is concentrated where it is needed for traction; (3) the same effect is obtained as when obstruction segments are used, but the drop with its attendant wear is eliminated; (4) telescopic screens allow the upper tumbler to be placed lower than with the cylindrical screens, as the tables at the rear end of the dredge can be placed higher. The advantages of the cylindrical screen are: (1) All of the screen plates are interchangeable and therefore a smaller number need be carried in stock; (2) it is less complicated; (3) it can be made more rigid; (4) a smaller proportion of the material goes on the upper tables, which is an advantage as with either type the upper tables have more than their share of material.

The screen structure should be designed so that it will take the torsional stress of the drive and also the bending movement without depending on the screen plates. This is sometimes accomplished by riveting braces between the longitudinal angles (Pl. XLI, *C*), but a simpler screen is obtained by so casting brackets on the tread rings that fit around the angles or bars as to take the shear from the rivets or bolts. Another method that accomplishes the desired result is to use sufficiently large gusset plates between the angles and the shell plates. All angles and plates should be held together with fitted bolts and spring washers instead of rivets, as the latter work loose too rapidly.

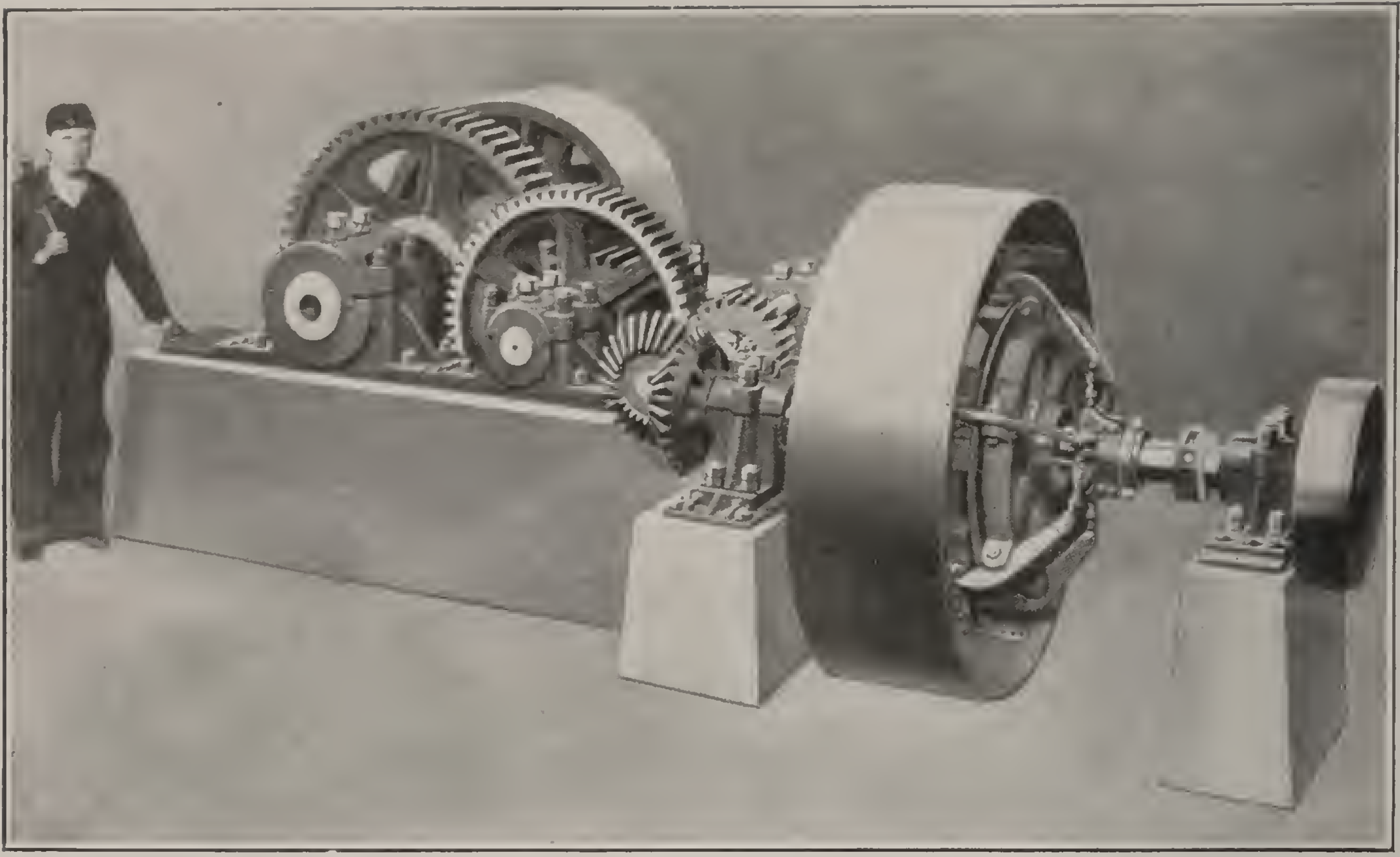
Screen plates are made either of high-carbon rolled plates with the holes drilled or of cast manganese steel. The holes should taper about one-eighth inch to every one-half inch of thickness of plate, and the small end should be placed inside the screen so that the holes will not clog. Longitudinal bars should be placed about 9 inches apart to carry the material with the screen as it revolves, thus spreading the gravel over a larger area. The bars cause the falling material to wear the screen plate more rapidly between the bars than directly under them. After such wear has developed the bars should be moved to cover the worn parts and expose the unworn parts.



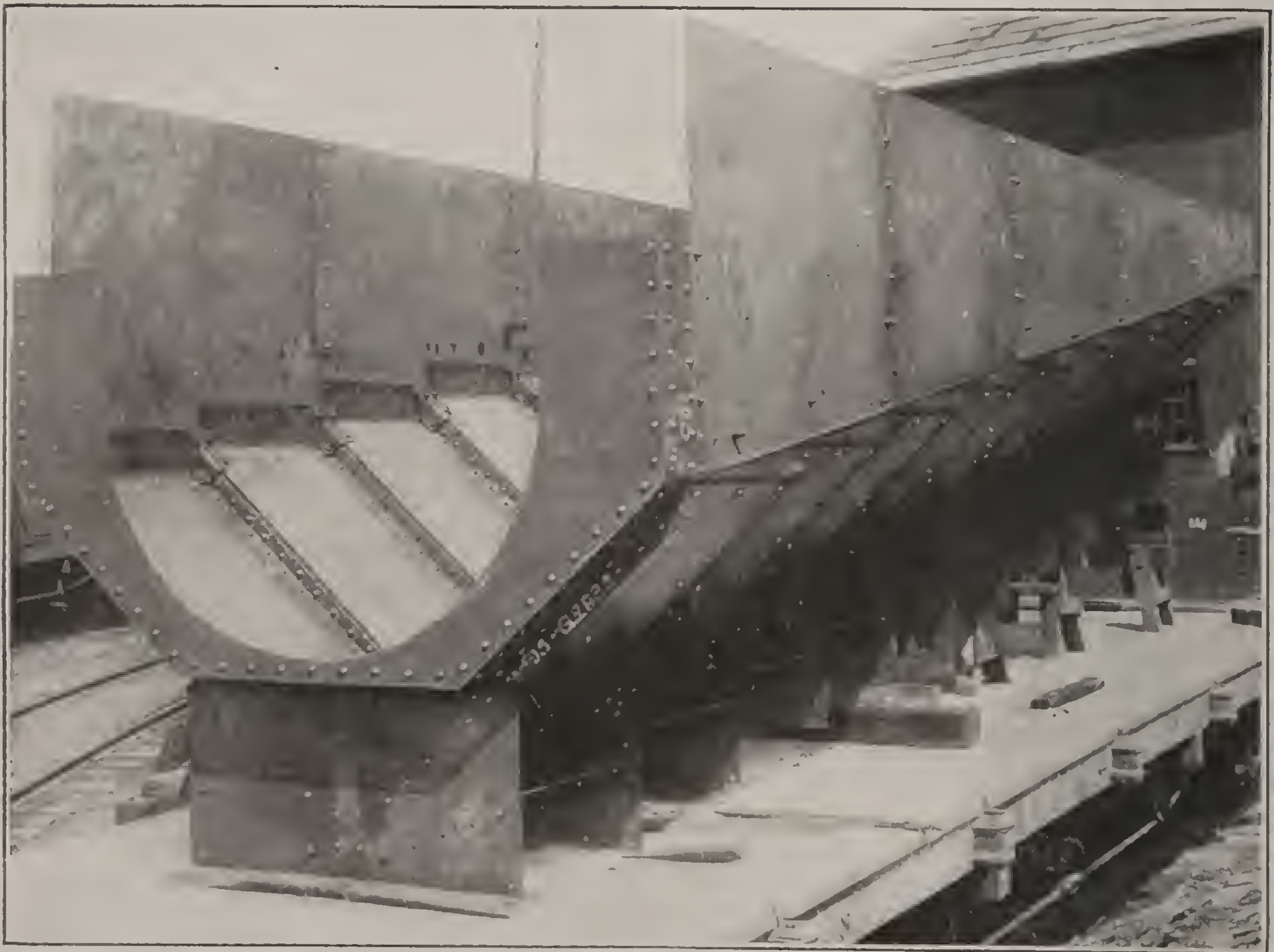
A. HOUSING AND DISTRIBUTOR FROM ABOVE, YUBA NO. 13 DREDGE.



B. MANGANESE-STEEL SCREEN PLATES, 90 INCHES LONG, 37 INCHES ACROSS, WITH 54-INCH RADIUS.



A. SCREEN DRIVE, SINGLE-ROLLER TYPE.



B. SCREEN HOUSING AND DISTRIBUTOR.

An obstruction segment is usually placed at the end of each screen plate. These segments are to keep the material from rushing through the screen too rapidly. The same result would be accomplished by reducing the pitch of the screen. One disadvantage of the segments is that the material is banked behind them and an uneven distribution along the screen results. Another objection is that there is excessive wear where the gravel drops over the segment. It is advisable to place one circle of bars at the upper end of the screen to retard the flow of the gravel as fed from the spout, also a set at the lower end to arrest the rapid flow of water to the conveyor belt.

The liners for the upper and lower shell plates should be bars about 6 inches wide by 1 inch thick rather than circular plates, as the former are more economical and easier to handle. Cast bars are superior as they can be made thicker where the greater wear comes.

For washing the gravel there are two methods, both of which have their advantages. In one method a large perforated pipe running the length of the screen is used. The diameter of perforations formerly used was about one-half inch, but this size proved too small as the holes often became clogged; now nozzles of 1 inch to 2 inches diameter spaced every 12 or 16 inches are used. It is impossible to support the pipe except at the two ends, and occasionally a pipe breaks. The other method is to place at each end of the screen a header with 2-inch nozzles, which direct the streams to the center of the screen. The advantage of the header is that it leaves the screen free from obstruction and more easily accessible. The nature of the gravel should determine which method of washing should be used.

The latest practice in driving the screen is to drive from the lower end principally because the driving machinery is then more accessible. The upper-end drive (Pl. XXXIX, *B*, and XLII, *A*) has, however, the advantage that the greater weight is concentrated at the upper end, thus producing better traction and less torsional strain.

It was formerly the custom to drive the screen from two rollers set at 45°. However, as the rollers did not wear down equally, there was a resultant slip, causing rapid wear. The newest dredges are designed with a single-drive roller (Pl. XLIII, *A*), placed directly under the center line of the dredge, or rather a few inches to one side of the center line. Plates XLII, *A*, and XLIII, *B* show two views of screen housing.

The idler rollers, or those places under the screen at the end opposite from the drive, are placed on a line 45° from the center line. By "cutting" these idler rollers or, in other words, shifting them out of line one way or the other the screen is kept from moving too far up or down. Unless these rollers are adjusted properly the thrust roller, which is usually placed at the lower end of the screen, is soon worn out. It is important that the tread rings be kept free of sand and grit. This is usually accomplished by splash rings, but it is well to have in addition a spray of water on the tread rings. Drive rollers and thrust rollers should be of special steel, either nickel-chrome or manganese. Tread rings should be of nickel-chrome.

Screens should have a peripheral speed of 150 feet per minute. The structure on which the screen is supported should be of steel as it is impossible to keep the bearings in line with a wooden structure.

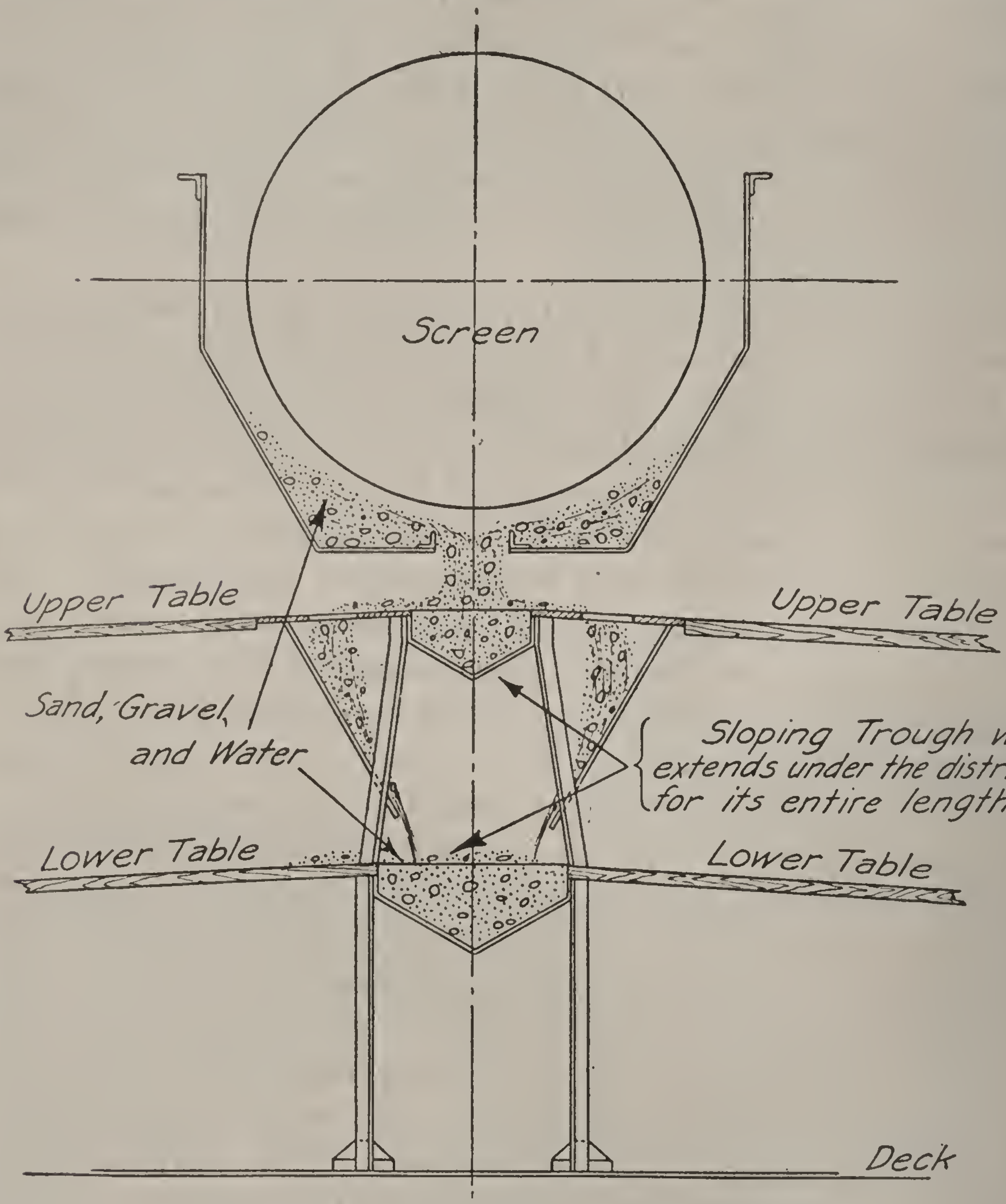
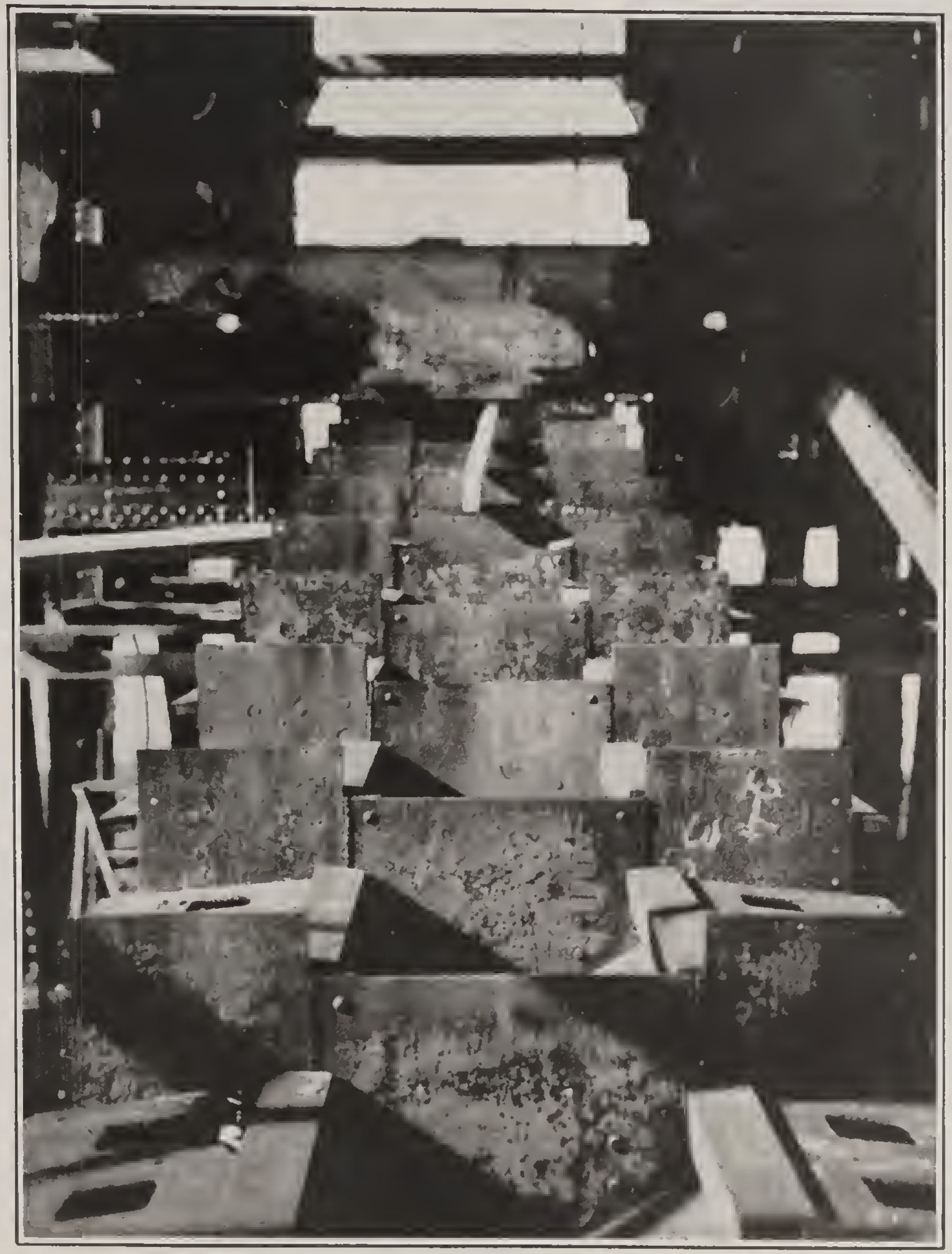
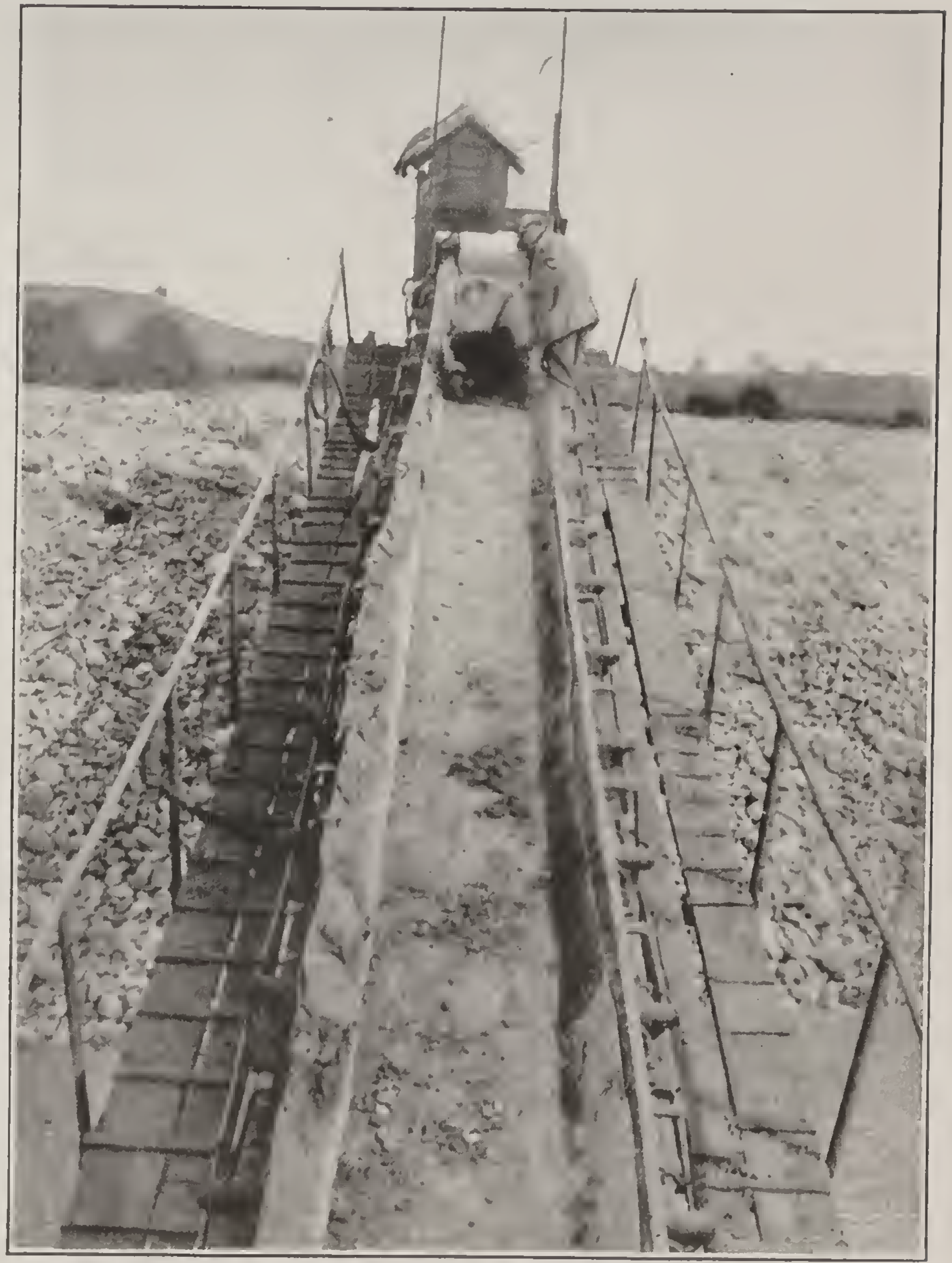


FIGURE 9.—Cross section of new type of distributor.

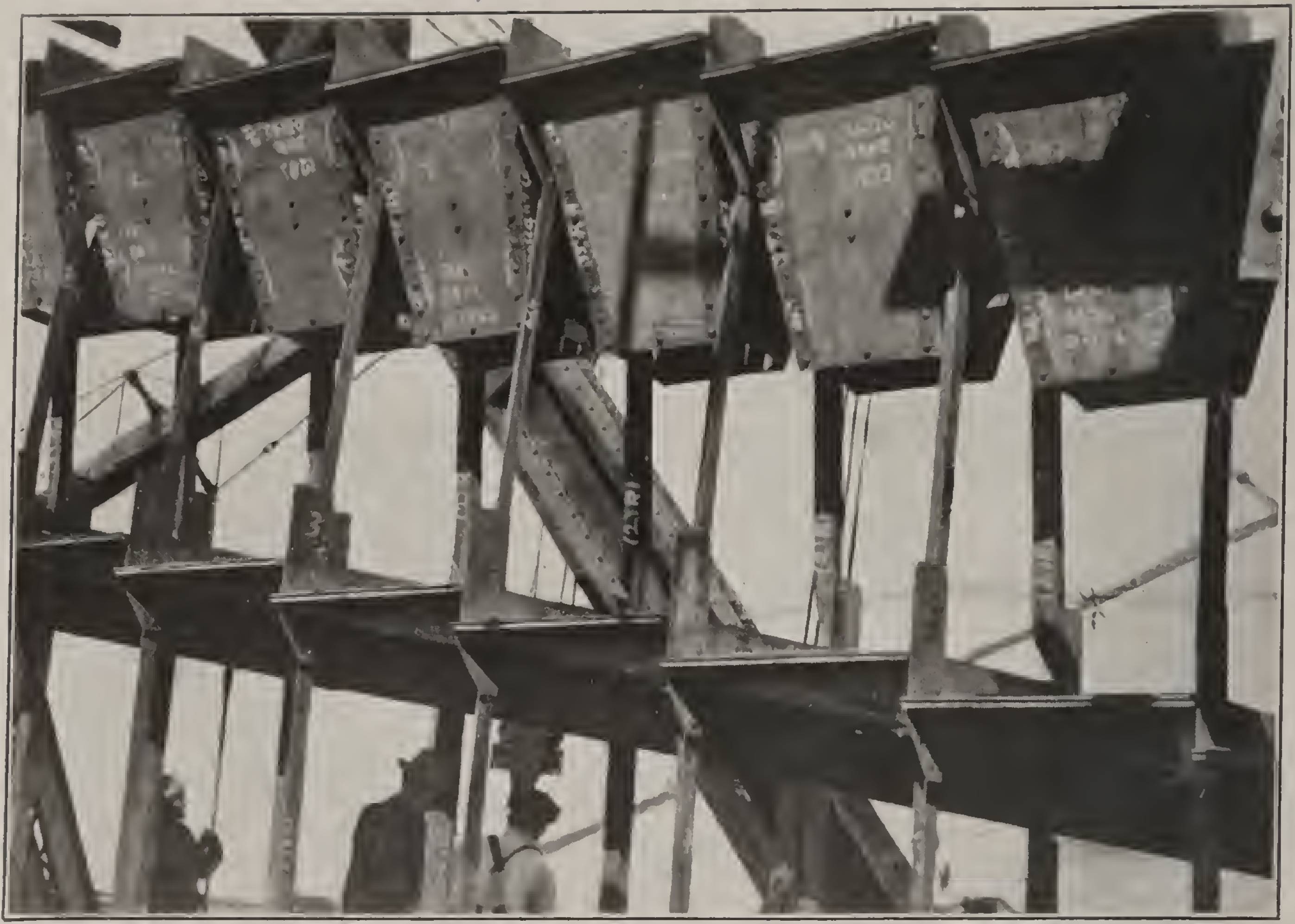
It is advisable to drive the screen with a variable-speed motor to eliminate as much jar as possible when starting. A silent chain drive is better than a belt drive. It is well to be able to reverse the direction of rotation of the screen as the wearing parts will last considerably longer, all holes and bars wearing according to the direction of rotation.



A. TOP OF DISTRIBUTOR.



B. FLAT TYPE OF STACKER BELT; NO TROUGHING IDLERS ARE USED.



A. SIDE VIEW OF DISTRIBUTOR, YUBA DREDGE. TABLES NOT IN PLACE.

B. SIDE VIEW OF DISTRIBUTOR. TABLES IN PLACE.

DISTRIBUTOR.

The object of the distributor is to catch the fine material and the water as they come from the screens and distribute them to the different sluices. A large proportion of the fine material is discharged at the upper end of the screen, and it is necessary to arrange the outlets of the distributor so that this material will be carried toward the lower end and in equal quantity to each of the tables.

Most of the material from revolving screens is discharged to one side of the center line, so that more goes to the tables on one side of the dredge than to those on the other side, unless the distributor is designed to equalize the amount fed to the tables.

Several types of distributors have been tried, but the one shown in Plate XLIV, *A*, was the standard type for some years. The bottom of this distributor was made of short eyebeams placed end to end, each about 6 inches lower than the one directly above. An opening was placed on each side with a sliding gate to regulate the feed. It was found impossible to get uniform feed to the different tables with this type, and another very serious objection was that it was very difficult to replace worn-out plates.

A new distributor (fig. 9) evolved by the Yuba Consolidated Goldfields has proved satisfactory. The bottom is separate from the sides, thus giving greater accessibility for repairs. It is a sloping trough with partitions placed every 2 or 3 feet. The trough fills with sand, which protects the steel bottom from wear. The material discharges over the sides of the trough onto the tables. When there are two decks of tables, openings are left at the head of the upper bank, and approximately half of the material drops to a similar distributing trough below. Plate XLV and figure 9 illustrate this feature.

STACKER.

At present the belt-conveyor tailing stacker is in general use, having practically superseded the bucket conveyor of the early dredges. There are not more than two or three dredges in the United States with bucket conveyors and the operators of these few dredges are the strongest advocates of the belt type. While this method of handling material was being developed by dredge operators, other industrial plants such as ore mills and cement works were proving the the belt-conveyor method of elevating and conveying broken rock or like substances to be much superior to any bucket system when the conditions were similar to those in dredging.

The belt stacker has the following advantages: (1) Lower cost of operation, as it does away with the innumerable pins and joints that are a source of wear and breakage, the breaking and bending of buckets, etc.; (2) less lost time owing to breakage; (3) less vibration, smoother running, and therefore less wear on the motor and the drive; (4) lighter weight; (5) with the belt stacker it is possible to dump the material at such height and distance behind the dredge as not to interfere with operations.

The stacking ladder consists of two parallel angle-iron trusses tied together at the bottom with braces, at the top with a continuous deck plate, and internally with cross braces. At the lower end the ladder is pivoted on a shaft placed as near as practicable to the receiving point.

It is important to have the drop from the screen chute to the conveyor placed as low as possible and it is necessary to place the pivot point near the receiving point so that the drop will not change as the ladder is raised and lowered.

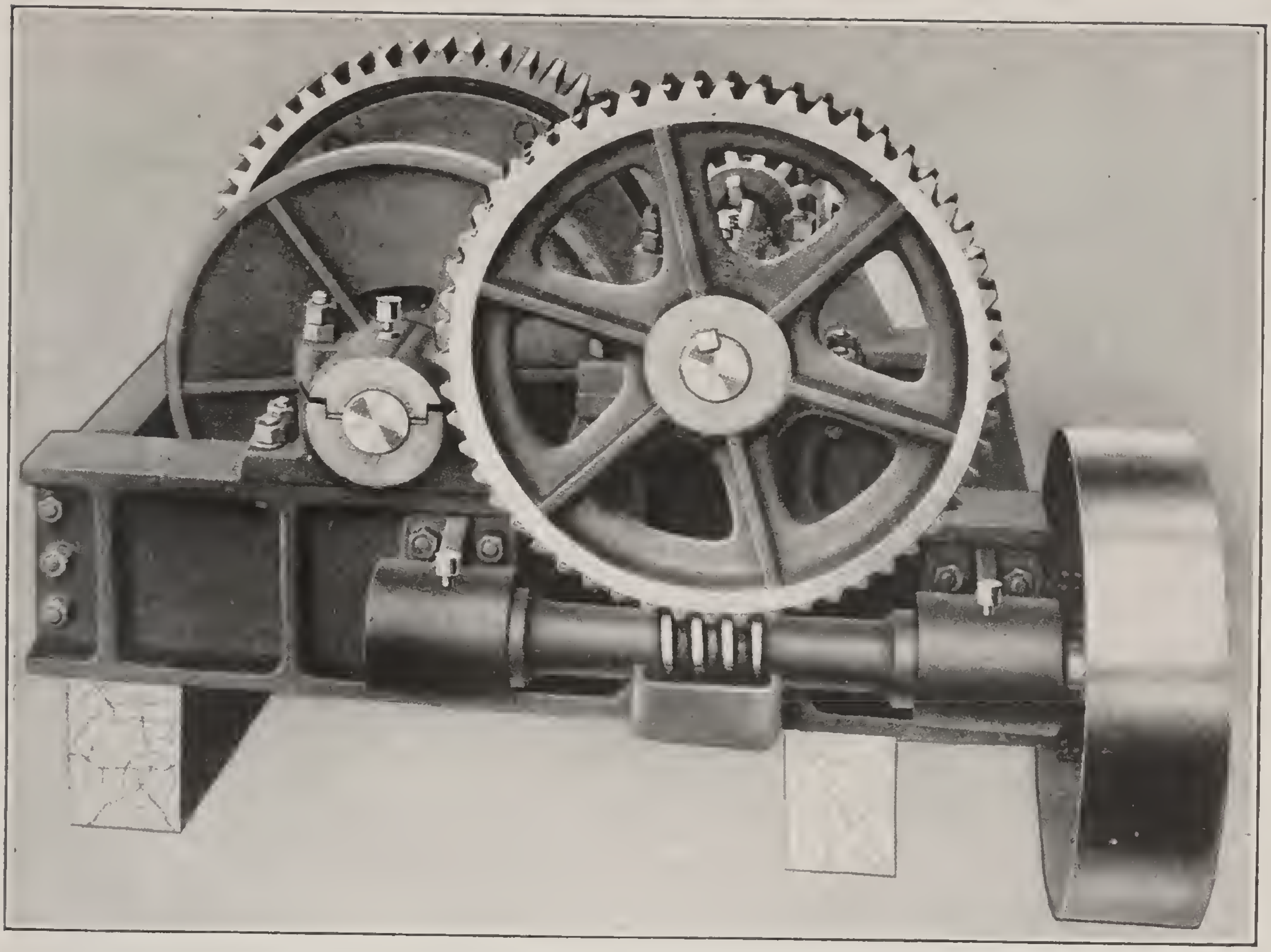
The outer end is supported and raised and lowered by wire rope from the stern gantry. A wire-rope sling, usually placed on the stacker, is arranged so that double ropes support the outer end and a single rope supports the center of the stacker ladder. It is best to do away as much as possible with steel rods and bars and to use wire rope, as the latter does not crystaliize so rapidly and a careful examination of the rope will determine whether the strains are too great, but a steel rod gives no indication of weakness before an accident. Several stackers have dropped owing to the breaking of bars, but when wire-rope slings have been substituted the trouble has been eliminated.

The stacker is hoisted by a small winch (Pl. XLVI, *A*) and the drive is through a worm wheel. No brakes need be used to hold the stacker in place, as the teeth are so designed that the stacker will not unwind the drum of its own weight; the motor must be operated to change the position of the stacker.

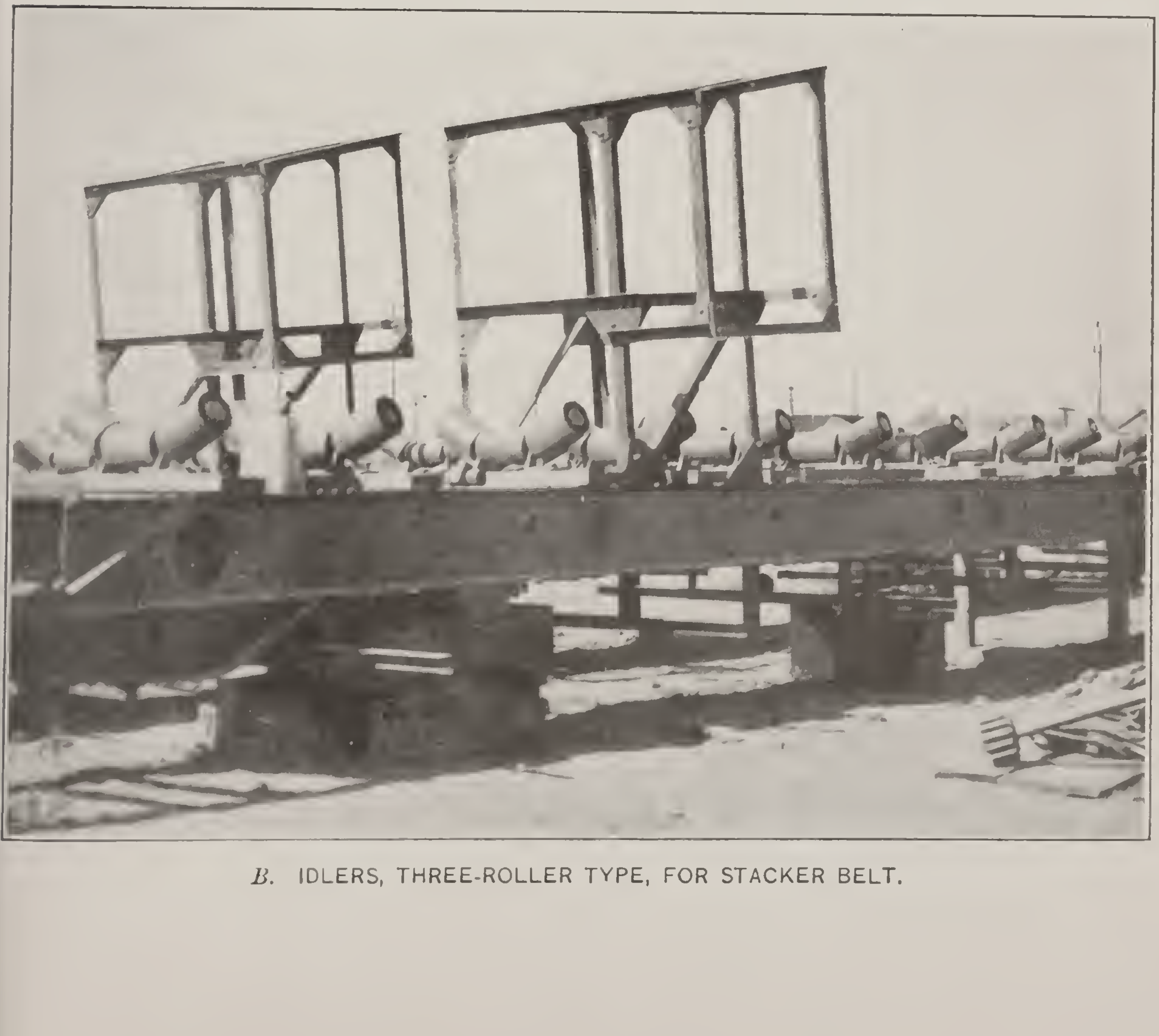
A recent innovation in the arrangement for distribution of the tailing from a dredge is shown in No. 16 dredge of the Yuba Goldfields. This dredge (Pl. XLVII) was designed with double-tailing stackers for reconstructing the channel of the Yuba River coincident with the recovery of gold from the gravel. With the exception of the stackers this dredge is practically the same as Yuba No. 15. The stackers are 225 feet long and the distance between the outer ends of the stackers is 416 feet.

BELTS.

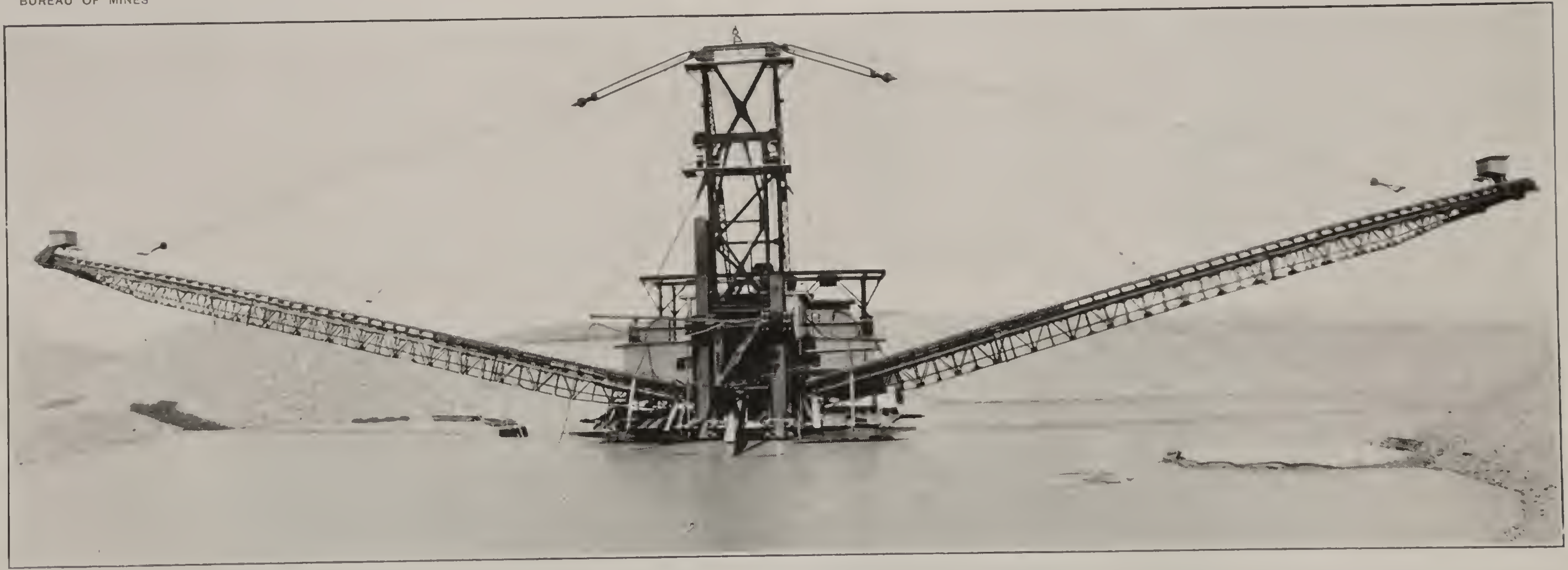
Practically all belting manufacturers have the duck used in their belting woven to specifications, some preferring a tight weave and others contending that a softer weave gives more flexibility and better wearing properties. One company states that the main body of the belt it manufactures is composed of duck of the best sea island cotton, and that experiments with different weights have shown that



A. STACKER WINCH, DOUBLED GEARED, FOR 9-FOOT DREDGE.



B. IDLERS, THREE-ROLLER TYPE, FOR STACKER BELT.



YUBA NO. 16 DREDGE, WITH DOUBLE TAILING STACKER.

a duck weighing approximately 32 ounces to the yard of 42-inch material gives the best satisfaction. Other firms use different weights of material.

Belts consist of three to eight plies of canvas, depending on the width and the length of the belt, covered on the carrying side and edges with one-sixteenth inch or one-eighth inch of rubber and on the other side with one thirty-second inch to one-sixteenth inch of rubber. The greatest wear occurs along a strip about 8 inches wide in the middle of the belt and in the past this part of the belt has often worn out while the remainder has remained in good condition.

Several methods of reinforcing the middle have been tried and many are in use. Figure 10 show the types now in most general use. Each of these has advantages and disadvantages. In figure 10, A shows the original construction, and B the first method of overcoming wear. Another serious difficulty experienced is that when the rubber is worn or punctured the water gets into the canvas which rapidly deteriorates, especially in cold climates where the moisture freezes and forces apart the canvas. The belt shown at C is known as the "Cleaveland construction." If the outer rubber is cut the damage does not reach to the inner layer of rubber and the efficiency of the belt is preserved.

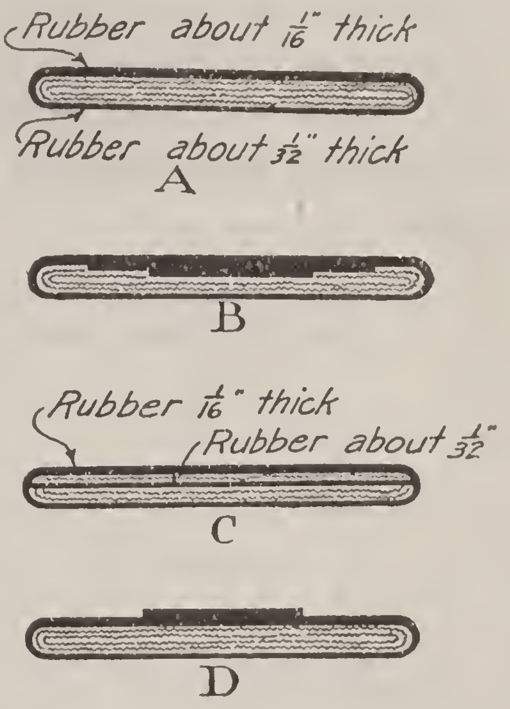


FIGURE 10.—Cross sections of four types of conveyor belts in general use. A, original type; B, first method of reinforcing center; C, "Cleaveland" construction; D, another method of reinforcement.

When a belt is not running true, owing to the idlers not being set correctly, the edges wear rapidly, hence it is well to continue the thick rubber well around the edges as shown at D.

During the early years of gold dredging in this country, considerable difficulty was found in procuring a conveyor belt that would wear satisfactorily. Many different materials were tried, and some "patent" belts lasted only a few hours when subjected to the severe wear on a gold dredge. At first it seemed difficult for the manufacturers to turn out a uniform product and belts bought from the same firm and subjected to the same wear would occasionally give entirely different service. In those days it was customary to buy belts according to specifications, stating the number of plies and weight of duck used, the friction pull and the thickness of the rubber covering, etc. Gradually competition among the various belt-making firms caused a life guaranty of the belts to be given and it is upon this basis that belts are now purchased. For dredges that have been at work for some time, an average is taken of the life of belts previously used and in new fields some time basis is generally agreed upon by the operator and the saleman. This method has proved satisfactory to the dredge operators. When the life of the belt exceeds the guaranty a bonus is sometimes paid for the extra service obtained though this practice is not universal. Occasionally payment has been arranged on an operating-hour basis and again on the number of yards handled during the life of the belt. On one 6-foot dredge payment was made on the basis of 20 cents an hour for total operating hours.

The life of standard conveyor belts in California varies from 9 to 14 months. On one dredge three belts gave unusually uniform wear of 458, 468, and 485 days, respectively. A fourth belt lasted 256 days only and adjustment for this short life was made by the manufacturer.

The size of the conveyor belt is determined both by the quantity of material it has to carry and by the size of the bowlders. Therefore, it does not vary only with the size of the bucket. The following table gives the width of belt usually used with buckets of different sizes.

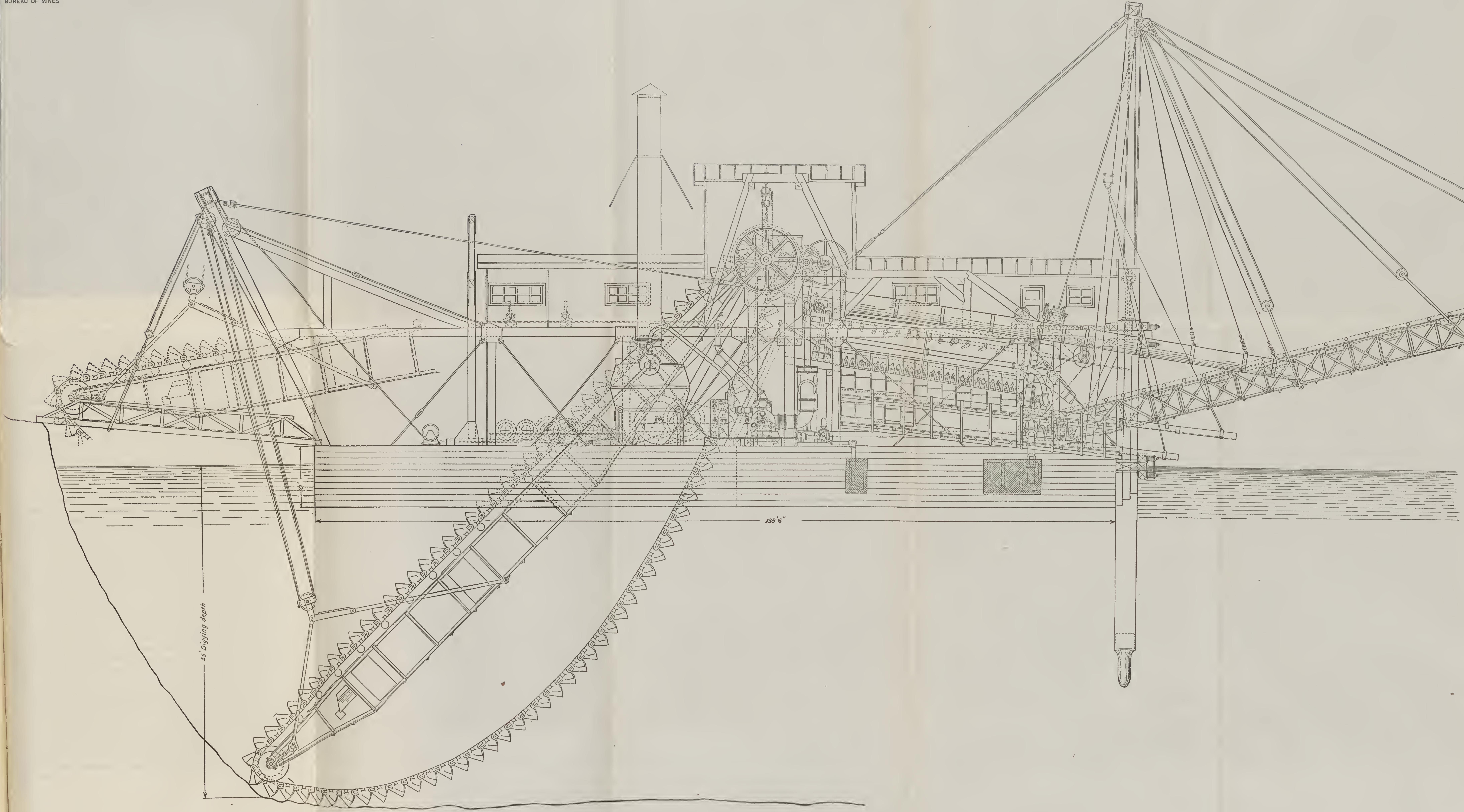
Width of conveyor belt used with different sizes of buckets.

Capacity of bucket, cubic feet.	Width of conveyor belt, inches.
3	24
4	28
5	32
6	32
7	32
9	36
13	36
16	42

When the gravel is of an unusual character the size of belt is determined by the size of the largest bowlders, as the chute feeding the belt should be narrower than the belt. The widths of belts corresponding to the sizes of gravel are as follows:

Width of belt, inches.	Size of largest pieces.
20	7-inch cubes.
24	9-inch cubes.
30	12-inch cubes.
36	15-inch cubes.
42	18-inch cubes.

When the size of the largest pieces determines the width of the belt, the latter should be run at the lowest possible speed that will carry the material, so as to insure minimum wear. When the tailings are small it is best to use a small belt and to run it at a high speed, not exceeding 400 feet a minute.



ELEVATION OF EQUIPMENT ON 8-FOOT NAMBULOA DREDGE.

Formerly the driving motor was directly connected through a pinion to the countershaft. Even with a cut pinion the noise and wear were excessive, as maintanance of correct alignment was impossible. The next method of applying power from the motor was by belt, and most conveyors are at present so driven. The disadvantage of the belt drive is that the necessary exposure to the weather seriously affects the belt. The stacker belts on the latest dredges are driven by a silent chain which combines the advantages of the other two methods and does away with the disadvantages. The loss in time is much reduced over the other two methods. There is practically no loss of time from the chain drive if the chain is kept well lubricated and is replaced when worn.

WIRE ROPES.

The expense of upkeep of the wire ropes is necessarily large on a dredge but on many dredges much expense can be obviated by the selection of sheaves of proper size. The rapid wear of ropes running on small sheaves is shown by the difference in the life of the bow swing lines on the same dredge. The port bow line usually runs over two or three more sheaves than the starboard line as the winch is placed on the starboard side of the dredge, and the port line often lasts only one-half or one-third as long as the starboard line.

The diameter of the running sheave, if possible, should be at least forty times the diameter of the rope (Pl. XXXVIII, *C*, p. 72); thus, for a 1-inch rope a sheave having a diameter of 40 inches should be used. It is of course impracticable with some equipment to carry out this proportion. For example, shore sheaves of such a size would be too heavy for efficient handling. However, on many dredges on which large lines and small sheaves are used the life of rope would be lengthened if the rope were smaller. This factor is less important in standing ropes, or those used as guys. The strength of a rope where it bends around the sheave diminishes directly as the size of the sheave is decreased, that is, the bending stress around a 12-inch sheave is twice as much as around a 24-inch sheave

There is not much difference of opinion among operators regarding the most desirable types of rope to perform the different operations but they do differ as to what quality is advisable—when the most expensive ropes should be used, and when less expensive lines are economical. The following table shows average practice as to ropes, "plough steel" designating the best rope made by the different manufacturers.

Data on average practice as to wire ropes used on dredges.

Where used.	Grade.	Number of strands.	Number of wires.
Ladder hoist	Plough steel	6	19
Bow swing	do	6	19
Stern swing	Crucible cast steel	6	19
Spud hoist	do	6	19
Stacker hoist	do	6	19
Stacker sling	do	6	7
Bow gantry guys	Galvanized bridge cable.	6	(a)

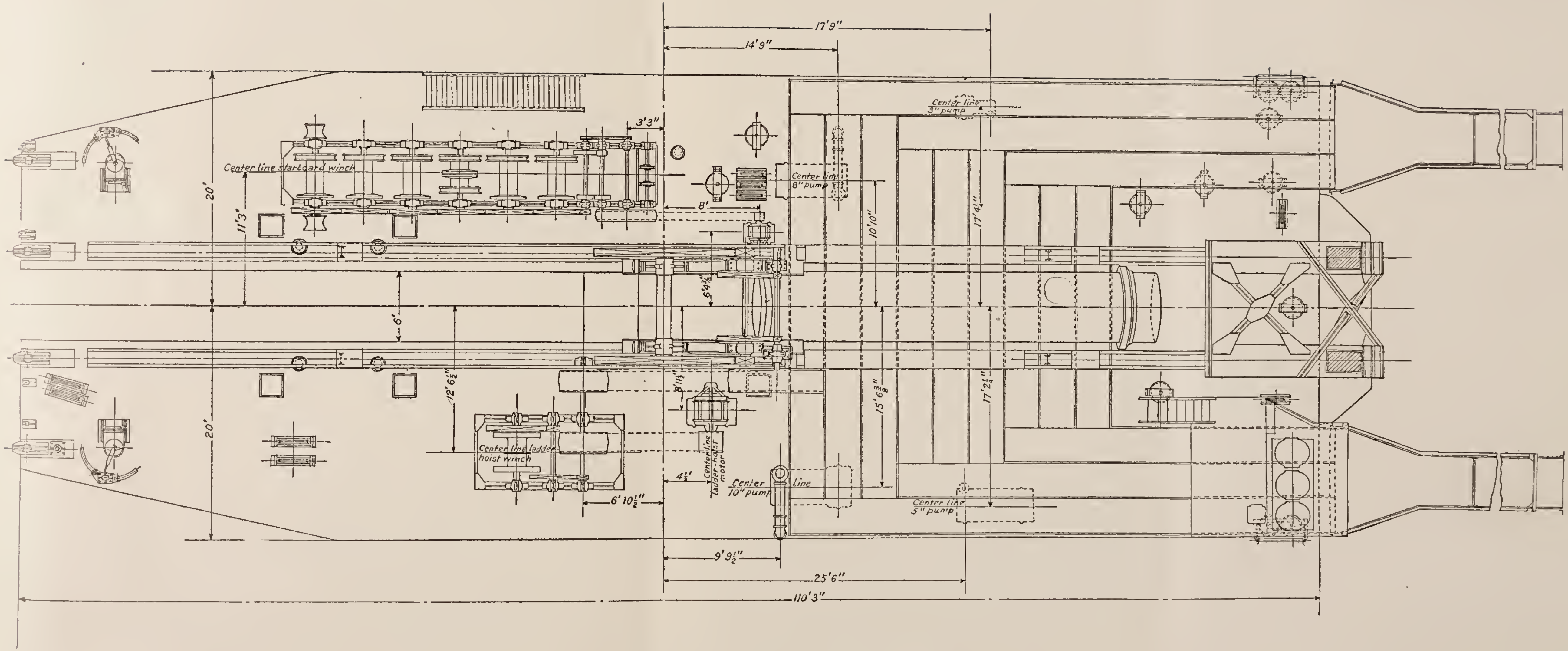
a Wire center.

Almost as important as the selection of the proper kind of rope is the care given it in use. A proper lubricant, of which there are a number on the market, should be freely employed and will increase the life of the rope.

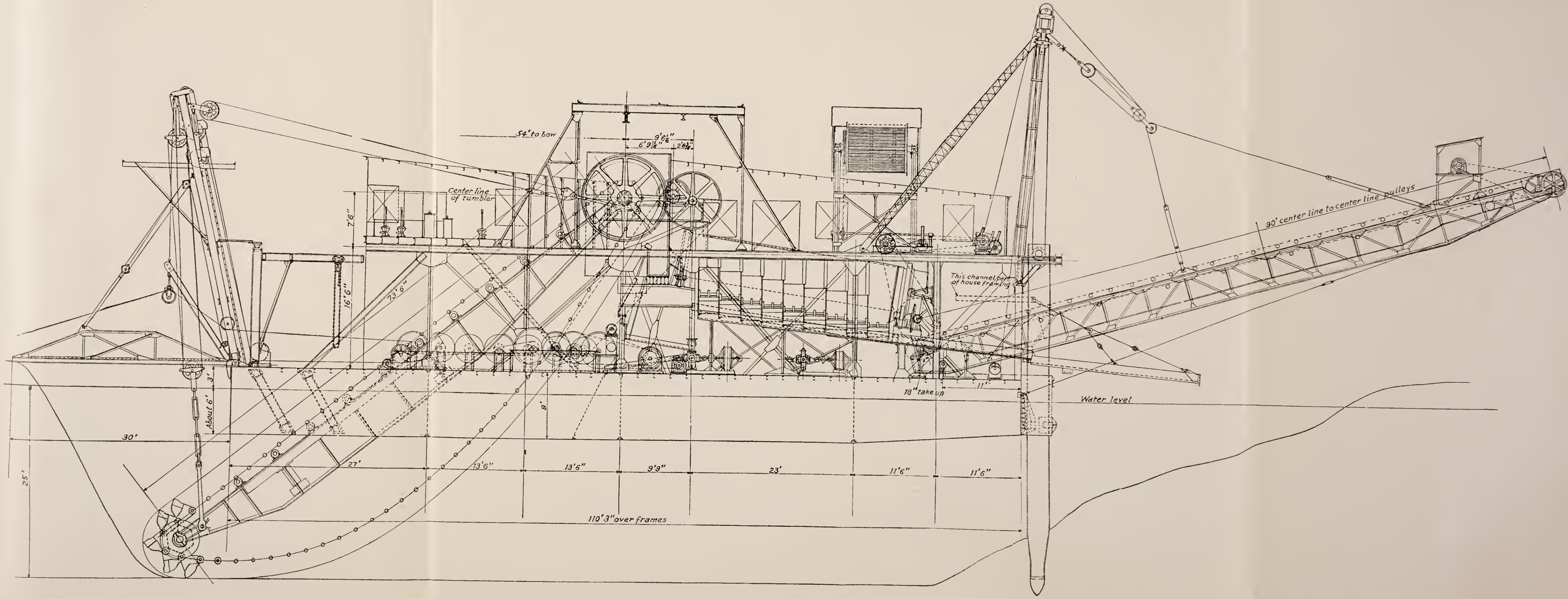
Manganese sheaves are easier on the ropes as they wear smoothly and in the grooves there is an absence of the ridges so common on cast-iron sheaves after they have become worn. Each sheave should be ground in the bore and a bronze or an iron bushing should be pressed in. Cast-steel sheaves are preferable if they are not subjected to wear.

The use of a socket on a rope in a permanent position is good practice provided the rope is properly put in place. The end of the wire should be spread out and washed with muriatic acid, then placed in the socket and zinc or spelter poured in. Lead or babbit should not be used nor should the wire be bent back. The object of using zinc or spelter is that either fuses to the wire, whereas lead depends upon the friction between it and the wires for holding power. Failure to fasten a rope properly in the socket has been the cause of several serious accidents.

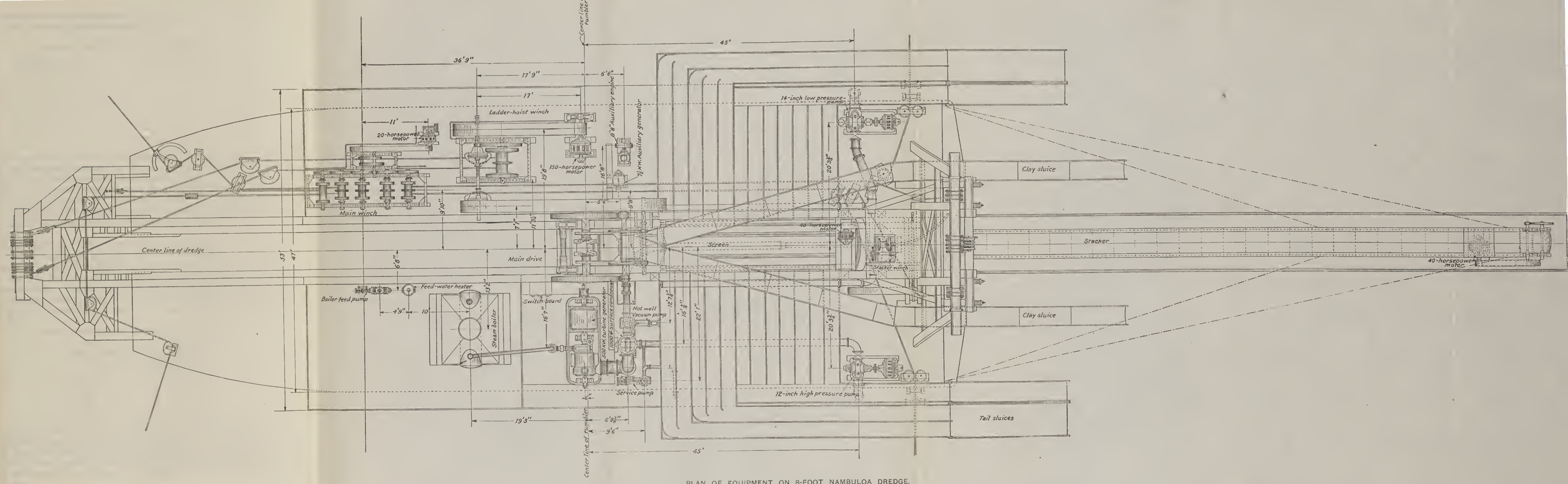
Different makes of wire rope are used for dredging and a number of the rope manufacturers supply a special brand for the purpose, 8,000 to 11,000 feet of rope being required for one of the large dredges. The following list gives the ropes furnished by one company for use on the Conrey No. 4 dredge, a total of more than 10,000 feet being required.



DECK PLAN OF A 7½-FOOT DREDGE.



ELEVATION OF A 7½-FOOT DREDGE.



PLAN OF EQUIPMENT ON 8-FOOT NAMBULOA DREDGE.

The less the angle of incline of the stacker, the greater the capacity of a belt, according to the following table:

Relation of angle of incline to capacity of conveyor belt.

Angle of incline, degrees.	Per cent of capacity of horizontal conveyor.
Horizontal	100
6	99
8	98
12	97
15	96
16	95
18	93
20	90
22	86

This loss in carrying capacity is caused by the material slipping on the belt; similarly the wear on the belt increases with the angle of incline. The incline should never be more than 22°. The above table is for material comparatively dry, and the loss of capacity increases rapidly with the increase of water. It is therefore important that as little water as possible be allowed to go through the screen to the stacker, and the screen should be arranged to take care of this.

Plate XLIV, *B* (p. 76), shows a stacker belt in use.

The following table gives the number of plies for the different widths of belt and the diameter of the stacker pulleys that should be used. It will be noted that the sizes given are larger than those generally used. A larger pulley will not pull more than a small one, as the pulling power depends only on the arc of contact. However, the pull is increased by facing the driving pulley with rubber or leather.

Number of plies required for different widths of stacker pulleys.

Width of belt.	Number of plies.	Diameter of upper drum.	Diameter of lower drum.
Inches.		*Inches.*	*Inches.*
24	5	30	26
28	5	34	28
32	6	38	32
36	7	42	36
42	8	48	42

The upper and lower drums should be large enough to reduce to a mimimum the strain on the fibers of the belt. The thicker the belt the larger the drums should be. The longer the belt, the greater should be the number of plies, but the number of plies given in the above table will be sufficient for any length of stacker used on gold dredges.

The snubbing rollers should be eliminated whenever possible but should be made as large as practicable when their use is necessary. The idlers should be placed near enough together so that there will not be too much sagging of the belt between idlers, as this causes internal wear (see Pl. XLVI, *B*, p. 78). For small belts the idlers are generally placed about 4 feet between centers, and for large belts they are placed 3½ feet apart. The idlers should not be supported on wooden planks, as is the custom, as these planks rot and do not hold the bearings securely. They have to be replaced in time and they also cause the spilled gravel to collect around the idlers and wear them rapidly, sometimes keeping the pulleys from revolving, so that the belt has to slide over them.

The trouble mentioned could be avoided by doing away with the flat deck plate that covers the stacker and substituting a sloping plate, high at the center, which would allow the spill to run into the pond. This arrangement would eliminate much labor at present necessary in scraping off accumulated material.

The return idlers underneath the stacker are made straight but instead of using a number of short pulleys mounted on a shaft, as is done in the usual belt-conveyor installation, one long pulley should be used. The short pulleys present a number of edges which are likely to cut into the belt. The return idlers are placed about 8 or 10 feet apart.

GUIDE ROLLERS.

Guide rollers should never be used to keep the belt from running off as their use injures the belt. If the head and tail pulleys are in line and the troughing idlers are set correctly the belt will keep to the center.

Rollers with a carrying surface that approximates a semicircle are generally accepted as the best type, although some operators prefer the spool-shaped rollers, making a flatter belt. The spool-shaped roller has a long center roll with larger rolls on each end which turn up the belt at the edges. The capacity of a belt running on spool-shaped rolls is not so large for a given width, but such rolls have the advantage of distributing the wear over a greater proportion of the width of belt than do the troughing type of rollers.

The stacker belt (Pl. XLIV, *B*, p. 76) should always be driven from the discharge end. Material conveyed on an inclined belt always slips somewhat at each troughing roller. The tighter the belt the less it slips. When the belt is driven from the lower pulley the carrying side of the belt is the slack side and as a consequence the belt wears faster and the carrying capacity is decreased.

Condition Nº1

Dredge in towing condition (light)

Center line of Lower tumbler 9'6" above deck

Items	Weight	Moment
Stationary items	1414175	124458200
Shifting items	201680	7854500
Displacement:	1615855	132312700

$\frac{132312700}{1615855} = 81.91'$

Center of buoyancy = 58.78

Center of gravity = 56.91

Arm = 1.87

Moment = 1615855 × 1.87 = 3015000'#

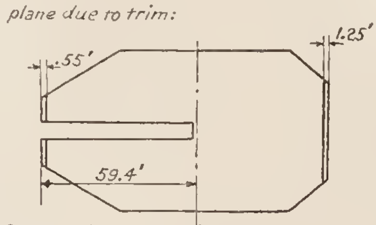
$H+h = \frac{I}{V} = \frac{4789120}{25980} = 185.0$

Mean draft = 5.11; h = 15.1

H = 185.0 − 15.1 = 169.9

$t\text{(at bow)} = \frac{3015000 \times 59.4}{1615855 \times 169.9} = 0.654$ (approx.)

Correction on account of change in water plane due to trim:



Corrected moment of inertia = 4632520 ft.^{4}

$H+h = \frac{4632520}{25980} = 178.5$; H = 163.4

$t\text{(at bow)} = \frac{1.87 \times 59.4}{163.4} = .68'$

$t\text{(at stern)} = .68 \times \frac{50.6}{59.4} = .58'$

Draft (at bow) = 5.11 + .68 = 5.79'

Draft (at stern) = 5.11 − .58 = 4.53'

Condition Nº2

Dredge in full working order

Ladder down; sand, water, and fuel on board

Items	Weight	Moment
Stationary items	1414175	124458200
Shifting items	201680	10934000
Stationary load	89100	9275000
Shifting load	22000	1230000
Displacement:	1726955	145897200

$\frac{145897200}{1726955} = 84.48$

Center of gravity = 59.48

Center of buoyancy = 58.86

Arm = .62

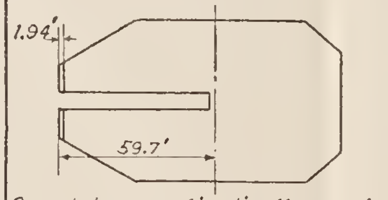
$H+h = \frac{I}{V} = \frac{4789120}{27323.5} = 175$

Mean draft = 5.40 h = 14.8

H = 175 − 14.8 = 160.2

$t\text{(at bow)} = \frac{.62 \times 59.4}{160.2} = 0.2295$ (approx.)

Correction on account of change in water plane due to trim



Corrected moment of inertia = 4656620 ft.^{4}

$H+h = \frac{4656620}{27323.5} = 170.3$; H = 155.5

$t\text{(at bow)} = \frac{.62 \times 59.7}{155.5} = .238$

$t\text{(at stern)} = .238 \times \frac{50.3}{59.7} = .202$

Draft (at bow) = 5.40 − .24 = 5.16

Draft (at stern) = 5.40 + .20 = 5.60

Condition Nº3

Dredge in full working order, digging its own flotation

Items	Weight	Moment
Stationary items	1414175	124458200
Shifting items	201680	7854500
	89100	9275000
	22000	950000
	1726955	142537700

$\frac{142537700}{1726955} = 82.54$

Center of buoyancy = 58.86

Center of gravity = 57.54

Arm = 1.32

H+h = 175 h = 14.5 H = 160.5

Mean draft = 5.4

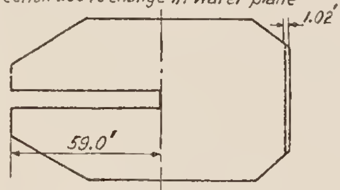
Moment (weights) = 1.32 × 1.726955 = 2280000

Moment (D.F.) = 25000 × 70 = 1750000

Total moment = 4030000

$$\left.\begin{aligned} t\text{(at bow)} &= \frac{4030000 \times 59.4}{1726955 \times 160.5} = 0.860' \\ t\text{(at stern)} &= .860 \times \frac{50.6}{59.4} = 0.735 \end{aligned}\right\}\text{approx.}$$

Correction due to change in water plane



Corrected moment of inertia = 4691120'^{4}

$H+h = \frac{4691120}{27323.5} = 171.5$; H = 157

$t\text{(at bow)} = \frac{4030000 \times 59.0}{1726955 \times 157} = .875$

$t\text{(at stern)} = .875 \times \frac{51}{59} = .755$

Draft at bow = 5.4 + .88 = 6.28

Draft at stern = 5.4 − .76 = 4.64

Condition Nº4

Dredge light, starting to dig above water line

Moment of D.F. = 25000 × 70 = 1750000'#

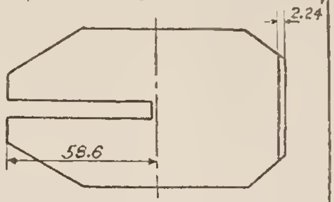
$t\text{(at bow)} = \frac{1750000 \times 59.4}{1615855 \times 169.9} = .39$

t (at bow) due to Condition No. 1 = 0.68

Total t (at bow) = 1.07 (approx.)

$t\text{(at stern)} = 1.07 \times \frac{50.6}{59.4} = 0.906$ (approx.)

Correction on account of change in water plane due to trim



Corrected moment of inertia = 4577120'^{4}

$H+h = \frac{4577120}{25980} = 176$ H = 160.9

Moment of weights = 3015000

Moment of D.F. = 1750000

Total moment = 4765000'#

$t\text{(at bow)} = \frac{4765000 \times 58.6}{1615855 \times 160.9} = 1.075'$

$t\text{(at stern)} = 1.075 \times \frac{51.4}{58.6} = .94$

Draft (at bow) = 5.11 + 1.08 = 6.19'

Draft (at stern) = 5.11 − .94 = 4.17

Note: The maximum draft of about 6'2¼" is obtained at the bow, when starting to dig above water, while dredge is in light condition.

Condition Nº5

Dredge digging 38 feet below water line; otherwise same as Condition No. 2

Moment of D.F. = 25000 × 70 = 1750000'#

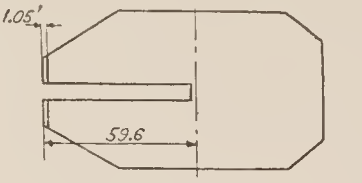
$t\text{(at bow)} = \frac{1750000 \times 59.4}{1726955 \times 155.5} = 0.388\ (-)$

t (at bow) due to Condition No. 2 = 0.238 (+)

Total t (at bow) = 0.150 (approx.)

$t\text{(at stern)} = 0.150 \times \frac{50.6}{59.4} = 0.128$ (approx.)

Correction on account of change in water plane due to trim



Corrected moment of inertia = 4717120'^{4}

$H+h = \frac{4717120}{27323.5} = 172.5$ H = 157.7

Moment of weights = 1069000 (+)

Moment of O.F. = 1750000 (−)

Total moment = 681000'#

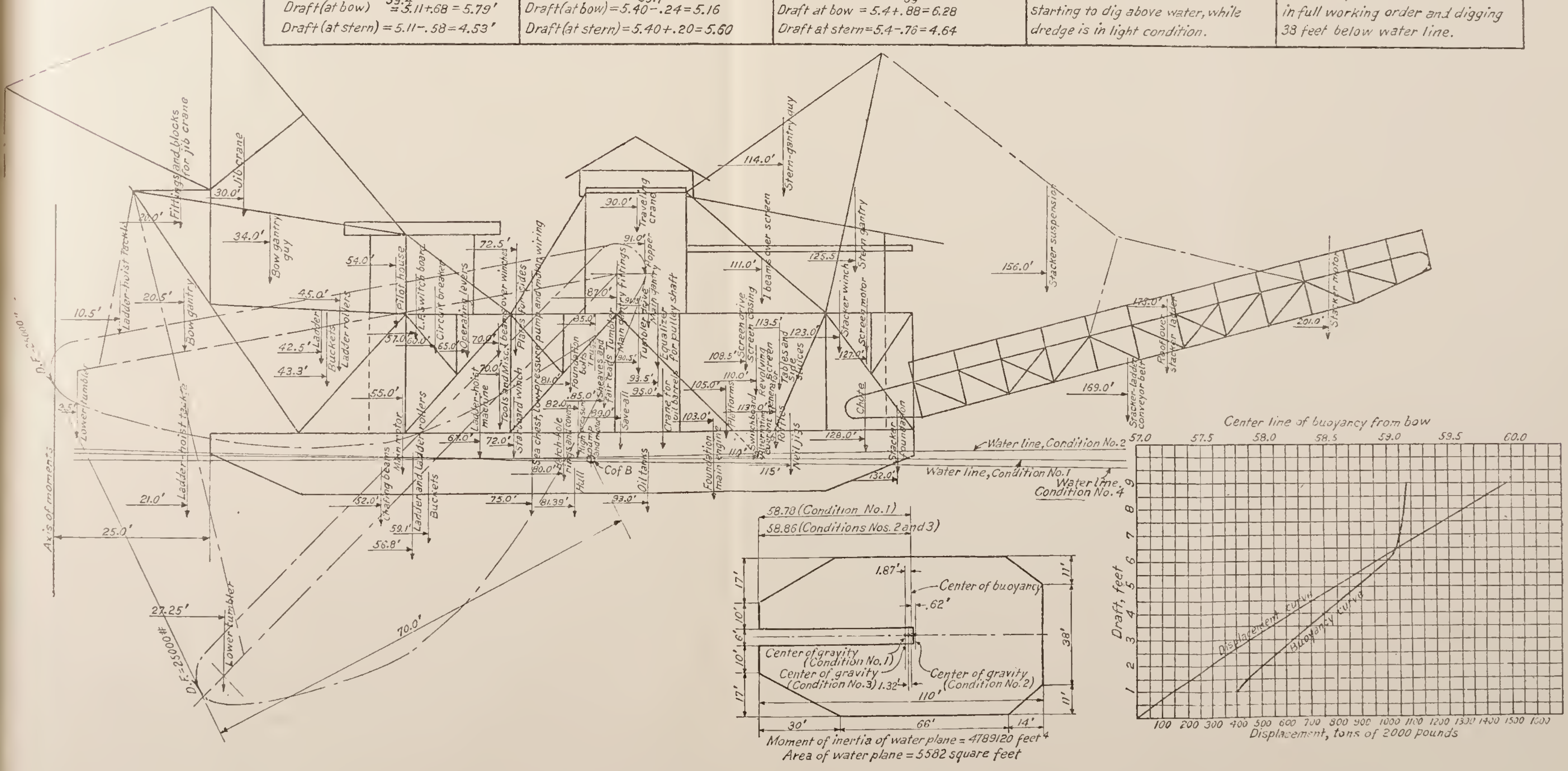
$t\text{(at bow)} = \frac{681000 \times 59.6}{1726955 \times 157.7} = .148$

$t\text{(at stern)} = .148 \times \frac{50.4}{59.6} = .125$

Draft (at bow) = 5.40 − .15 = 5.25

Draft (at stern) = 5.40 + .13 = 5.53

Note: The dredge is floating practically on even keel when in full working order and digging 38 feet below water line.



FLOTATION DIAGRAM OF A 5½-FOOT DREDGE.

Sizes and uses of ropes used on Conrey No. 4 dredge.

Number of pieces.	Dimensions of rope.[a]	Where used.
2	2½ by 19 by 195	Steam gantry guys.
2	2 by 19 by 375	Bow gantry guys.
1	2 by 19 by 200	Stacker suspension.
1	2 by 19 by 220	Do.
2	1½ by 19 by 1,100	Ladder hoist.
1	1⅜ by 19 by 46	Adjusting rope for stacker guys.
1	1½ by 19 by 500	Straps for side swing lines.
1	1¼ by 19 by 650	Starboard speed line.
1	1¼ by 19 by 655	Port speed lines.
2	1¼ by 19 by 600	Bow swing lines.
1	1 by 19 by 440	Stacker hoist rope.
2	1 by 19 by 400	Stern lines.
1	1 by 19 by 420	Stacker hoist.
1	¾ by 19 by 190	Gang-plank hoist.
1	¾ by 19 by 510	Side guys for stacker.
4	¾ by 19 by 80	Sluice-pan guys.
4	¾ by 19 by 60	Do.
8	¾ by 19 by 43	Do.

[a] The first figures indicate the diameter in inches, the second figures the number of wires to the strand, and the last figures the length in feet.

As the lengths of ropes vary on different dredges the following table of the most important ropes on Yuba dredges is given, a total of about 10,000 feet being used for all purposes.

Data regarding most important ropes used on Yuba dredges.

Where rope is used.	Diameter of rope.	Number required.	Length of each.	Total length.
	Inches.		*Feet.*	*Feet.*
Bow gantry guys	2½	2	239	478
Stern gantry guys	2	2	135	270
Tail-sluice support	½	2	250	500
Foot-bridge suspension	½	2	250	500
Slacker sling	2	1	172	172
Slacker guys	1	1	120	240
Ladder hoist	1½	2	1,200	2,400
Slacker hoist	1	1	600	600
Speed hoist	1	2	600	1,200
Bow swing lines	1¼	2	700	1,400
Stern swing lines	1	2	500	1,000

The life of rope depends on the quality, the use to which it is subjected, and the care it is given. The best brands of ladder-hoist ropes have lasted two years or over, whereas some rope in use for side lines has worn out in 90 days.

GENERAL DETAILS OF DREDGE CONSTRUCTION.

Plate XLVIII (p. 80) shows a plan and Plate XLIX an elevation of equipment on a 7½-foot dredge, showing the general arrangement of parts. Plate L shows a plan view and Plate LI a side view of the assembly of an 8-foot dredge. Plate LII shows a flotation diagram for a 5½-foot dredge. The table following presents general statistics regarding dredges used in Montana:

General statistics concerning dredges used in Montana.

Item.	Steam dredges.			Electric dredges.			
	Maggie Gibson.	Conrey No. 1.	Conrey No. 2.	Conrey No. 1.	Conrey No. 2.	Conrey No. 3.	Conrey No. 4.
Power used	Steam	Steam	Steam	Electricity	Electricity	Electricity	Electricity.
Date built	1896	1899	1901	1908	1908	1906	1911.
Type	Double-lift sluice	Single-lift sluice	Single-lift sluice	Single-lift table stacker.	Single-lift table stacker.	Single-lift table stacker.	Single-lift table stacker.
Anchorage	(a)	Lines	Lines	Spud	Spud	Lines	Spud.
Average monthly yardage worked, cubic yards.	27,000	41,360	52,520	96,270	62,709	82,415	300,000.
Hull dimensions:							
Draft	5 feet 3 inches	5 feet 4 inches	5 feet 9 inches	5 feet	5 feet 6 inches	6 feet	9 feet.
Length	110 feet	100 feet	110 feet	96 feet	102 feet	130 feet	150 feet.
Width	35 feet	44 feet	44 feet	44 feet	44 feet	48 feet	58 feet.
Depth	7 feet 6 inches	7 feet 6 inches	7 feet 6 inches	9 feet	9 feet	7 feet 11 inches	13 feet.
Pontoon dimensions:							
Length	30 feet	38 feet 6 inches	40 feet	None	None	50 feet	None.
Width	24 feet	35 feet	29 feet			40 feet	
Depth	4 feet	3 feet	3 feet 10 inches			3 feet 11 inches	
Stacker dimensions:							
Length	None	None	None	90 feet	85 feet	None	130 feet.
Width of belt				34 inches	34 inches		4 feet.
Motor drive				Upper end 30 feet	Upper end 30 feet		Upper end 55 feet.
Tailing height							
Dimensions of steel spuds:							
Length, feet	None	None	None	48	54	None	80.
Width, feet				3	3		4½.
Thickness, feet				2	2		3.
Weight, pounds				26,500	29,000		88,000.
Number				1	1		2.
Dimensions of wooden spuds:							
Length, feet	30	None	None	48	54	None	None.
Width, feet	1½			3	3		
Thickness, feet	4			2	2		
Number	2			1	1		
Horsepower:							
Bucket chain	45	60	73	100	100	150	550.
Trommel	5	6	8	30	30	10	100.
Pumps	40	79	147	160	160	175	285.
Winches	10	12	4	20	20	115	250.
Miscellaneous	None	19	16	70	70	10	50.
Total horsepower	100	176	248	380	380	460	1,235.
Ladder data:							

feet.							
Dimensions of tumblers:							
Upper tumbler—							
Distance above deck	12 feet 7 inches	24 feet 6 inches	29 feet 3 inches	25 feet 9 inches	29 feet 3 inches	31 feet	35 feet 6 inches.
Weight, pounds	2,000	2,000	4,050	8,970	8,970	8,970	16,000 to 25,300.
Number of sides	5	5	5	6	6	6	6.
Shaft diameter, inches	9	11	12	16	16	16	25.
Lower tumbler—							
Weight, pounds	3,000	2,500 to 3,075	4,600	5,800	5,800	5,800	18,000 to 19,800.
Number of sides	6	6	6	6 to round	6 to round	6 to round	6 to round.
Shaft diameter, inches	6	6 to $8\frac{1}{2}$	10	$13\frac{1}{4}$	$13\frac{1}{4}$	$13\frac{1}{4}$	$15\frac{3}{8}$.
Bucket-chain data:							
Type	Open link	Open link	Open link	Close connected	Close connected	Open link, later changed to close connected.	Close connected.
Drive	Sprocket	Sprocket	Sprocket	Belt	Belt	Gear	Gear.
Weight, pounds	40,000	104,000	125,000	169,000	225,000	297,000	357,500 to 415,300
Capacity of buckets, cubic feet	5	5 to $7\frac{1}{2}$	10	$7\frac{1}{2}$	$7\frac{1}{2}$	13 open, $9\frac{1}{2}$ closed	16 to 17.
Number in chain	35	41	41	60	80	43 open, 80 closed	80.
Weight, pounds	750	750 to 1,500	1,850	2,594	2,594	2,900	4,468 to 5,191.
Pitch, inches	26	26 to $29\frac{3}{8}$	$31\frac{5}{16}$	$32\frac{3}{4}$	$32\frac{3}{4}$	$32\frac{3}{4}$	40.
Buckets dumped per minute	7	5 to 11	5 to 11	16	15	15	18 to 22.
Type	2-web	2-web	3-web	3-web	2-web	2-web	2-web.
Material	Steel	Steel	Steel	(c)	(c)	(c)	(c).
Hood and base, number of pieces	2	2	2	2	(d)	(d)	1.
Links—							
Weight, pounds	300	770	900	None	None	1,250	None.
Pitch, inches	26	$29\frac{3}{8}$	$31\frac{5}{16}$			$32\frac{3}{4}$	
Number in chain	35	40	40			43	
Material	Steel	Steel	Steel			Steel	
Type	2-web	2-web	3-web			3-web	
Pins—							
Diameter, inches	3	3 to $4\frac{5}{8}$	$4\frac{7}{8}$	$4\frac{5}{8}$	5	5	7 to 8.
Weight, pounds	50	50 to 130	155	168	184	184	520 to 495.
Material	Steel	Steel	Steel	Steel	(c)	(c)	(c).
Type	Double lug	Lock pin	Lock pin	Single lug	Single lug	(e)	Single lug.
Pin bushings—							
Type	Half circle	Full circle	Full circle	Full circle	Full circle	Full circle	Full circle.
Material	Manganese	Manganese	Manganese	Manganese	Manganese	Manganese	Manganese.
Trommel data:							
Length	14 feet	16 feet	15 feet 2 inches	35 feet	35 feet	18 feet	48 feet 6 inches.
Diameter, inches	48	46	48	51 to 72	51 to 72	61	98.
Type	Straight cylinder	Straight cylinder	Straight cylinder	Stepped cylinder	Stepped cylinder	Straight cylinder	Straight cylinder.
Drive	Sprocket	Sprocket	Sprocket and bevel gear.	Belt	Belt	Belt	(f).

[a] Originally spud; changed to lines.
[b] Changed to 116 feet.
[c] Steel; later changed to manganese.
[d] Steel buckets in two pieces; manganese in one.
[e] Lock pin; later changed to single lug.
[f] Gear; later changed to belt.

General statistics concerning dredges used in Montana—Continued.

Item.	Steam dredges.			Electric dredges.			
	Maggie Gibson.	Conrey No. 1.	Conrey No. 2.	Conrey No. 1.	Conrey No. 2.	Conrey No. 3.	Conrey No. 4.
Trommel data—Continued.							
Plates—							
Thickness, inches	5/8	5/8	5/8	5/8	5/8	5/8	1.
Material	Steel	Steel	Steel	(*a*)	(*a*)	(*a*)	(*a*).
Perforations	4 by 4 inches	4 by 4 inches	4 by 4 inches	1/2 to 3/4 inch	1/2 to 3/4 inch	4 1/4 by 6 inches	1/2 to 3/4 inch.
Dimensions of tables:							
Grade, per cent	None	None	None	12 1/2	12 1/2	12	12 1/2.
Number				20	20	10	24.
Width, inches				30	30	30	30.
Length, feet				18	18	12	11 to 29 3/4.
Riffles (angles), inches				1 1/4	1 1/4	1 1/4	1 1/4.
Perforated plates						3/8-inch hole	
Pump data:							
Number	2	2	3	4	4	5	3.
Size and kind	4-inch water 12-inch gravel	10-inch water 14-inch water	10-inch water Two 14-inch water	3-inch water 8-inch water 12-inch water 6-inch sand	3-inch water 8-inch water 12-inch water 6-inch sand	3-inch water Two 10-inch water 12-inch water 14-inch water	4-inch water. 14-inch water. 16-inch water.
Make	Morris	Morris	McKay	Morris	Morris	Morris-Worthington.	Worthington.
Gallons per minute	4,000	6,500	8,100	6,700	6,700	12,500	12,000.
Main winch data:							
Length	12 feet	26 feet	26 feet	20 feet 1 inch	20 feet 1 inch	11 feet 10 inches	29 feet 10 inches.
Width	6 feet	16 feet	16 feet	4 feet 11 inches	4 feet 11 inches	8 feet 6 inches	7 feet 5 inches.
Diameter of drums, inches	16	16	16	16	16	30	24.
Number of drums	6	8	8	8	8	2	8.
Tail sluices:							
Grade, per cent	None	None	None	12 1/2	12 1/2	None	12 1/2.
Number				2	2		12.
Length, feet				40	40		17 to 44.
Width, inches				48	48		48.
Riffles (angles), inches				1 1/4	1 1/4		1 1/4.
Undercurrents:							
Grade, per cent	None	10	10	None	None	10	12 1/2.
Number		1	1			2	12.
Length, feet		40	60			54	7 1/2.
Width, inches		10	10			13	34.
Riffles—							
Material		Wood	Wood			Wood	Angles.
Size, inches		1 by 1 1/2	1 by 1 1/2			1 by 1 1/2	1 1/4.

Perforated plates:							
Width of slots, inches		1/16	1/16			1/16	1/4.*b*
Well save-all:							
Grade, per cent	None	8	8	8	8	8	8.
Number		1	1	1	1	1	1.
Length, feet		30	30	7½	7½	25⅝	18½.
Width, inches		12	12	18	18	18	18.
Upper flume:							
Grade, per cent	None	5	5	None	None	6	None.
Riffles—							
Material		Wood	Wood			Angles	
Size, inches		1¼ by 2	1¼ by 2			2 by 2½	
Length		21 feet	22 feet			28 feet 10 inches	
Width, inches		42	50			62	
Lower flume:							
Grade, per cent	5	5	5	None	None	6	None.
Riffles—							
Materials	Wood	Wood	Wood			Angles	
Size, inches	2 by 4	2 by 4	2 by 4			2 by 2½	
Length, feet	103	120	120			135	
Width, inches	38	52	52			66	
Total gold-saving area, square feet	320	656	692	1,231	1,231	1,264	3,000.

a Steel; later changed to manganese.

b Holes instead of slots.

SPECIAL STEELS FOR DREDGE PARTS.

PROPERTIES OF ALLOYS OF STEEL.

The gold-dredging industry has played an important part in the development of the different alloys of steel on account of the exceptionally severe service imposed. On the earlier dredges cast iron entered largely into the different machine parts, and the wearing parts were made of carbon steel. Cast iron has now given way to cast steel and all parts subjected to wear and those subjected to severe strains are made of one of the alloys. Thus, in order to design an efficient modern dredge, a thorough knowledge of steel and its properties is necessary.

MANGANESE STEEL.

Manganese steel has been the most important agent in bringing gold dredges to their present high efficiency. Forged or rolled manganese steel is little used on a dredge but there are many places where it could be used to advantage, especially in the shape of thin bars for lining riffles, etc. It has not yet been proven that forged manganese steel can take the place of cast manganese steel to any extent, so only the latter is here considered.

Manganese steel owes its importance to a combination of properties rather than to any one characteristic. It is not so hard as chilled iron; in fact, its ability to resist abrasion does not depend on its hardness alone but on its toughness and hardness and what is known as its ability to flow, and also on its ability to endure repeated distortions of the wearing surface without surface friction, which would result in constant loss of small particles. Its comparative hardness is shown by the scleroscope, cast steel being 30, manganese steel 50, and chilled cast iron 70.

Its property of flowing and its ductility must be guarded against when designing dredge parts; for instance, a bucket bushing must be made shorter than the back eye of the bucket, and clearance must be allowed all around wearing plates when there is a liability of their becoming wedged in place by undue expansion.

Manganese steel will withstand abrasion from heavy impacts much longer than will ordinary steel. Also, attrition caused by sliding material has less effect on manganese steel than on ordinary steel. Under certain conditions it does not resist the attrition of fine sands or slimes as well as other materials do; so manganese steel can not be said to be a panacea for all abrasive ills. It has high tensile strength—about 90,000 pounds per square inch for cast test pieces which have not been forged—but the elastic limit is low and not well defined, running about 40,000 to 45,000.

Partly because the metal is poured while very hot, it lends itself to the casting of intricate designs and thin sections, but exceptional care must be exercised in designing parts to avoid abrupt changes in the thickness of sections, for the castings are put through a severe heat treatment. The metal as cast is hard and brittle. Hence it is heated to about 1,700° F. and quickly plunged in cold water, which usually distorts the casting so that considerable restraightening is required. As manganese is a poor conductor of heat the sections of metal to be annealed can not be thick, not much more than 4 inches, as internal cracks appear in pieces with thicker sections.

Manganese-steel castings should never be heated for any purpose after they leave the foundry, for if they are heated to any other than the correct temperature they lose their valuable properties. Manganese steel differs from other steel in that the skin or outside is softer than the metal underneath, as it decarbonizes after annealing; and because manganese can be dented with a chisel it has sometimes incorrectly been thought inferior. The greatest drawback to a more general use of manganese steel is that it can not be machined with cutting tools, but such efficient grinding apparatus has been perfected that this objection is largely nullified. Formerly it was necessary to cast a soft-steel insert where boring was to be done, and this practice is still followed to some extent; however, it is usually better to grind the bore for all wheels. Manganese can readily be distinguished from other steels for, unlike them, it will not attract a magnet.

One of the peculiarities is that it wears much smoother than does ordinary steel, and its coefficient of friction is less. This may be an advantage or a disadvantage. For instance, on a dredge with a long bucket line if the heel plates are of manganese steel the buckets will sometimes slip to such an extent that undue vibration of the main drive will occur; such heel plates should be made of nickel-chrome steel or some other serviceable metal. It is desirable, however, that screen tread rollers and sheaves wear smooth, and this property is therefore advantageous. It is possible to cast manganese screen plates (Pl. XLII, *B*, p. 74) $\frac{3}{4}$ of an inch wide with $\frac{3}{8}$-inch holes. These wear several times longer than plates made of ordinary steel and are rapidly taking their place.

The best method of determining whether manganese is the proper material to use under certain conditions is to observe it in service. It has often proved unsuccessful where the conditions were thought to be ideal, and it has been found to suit admirably where little promise was held out for its success.

NICKEL STEEL.

Nickel steel takes the place of ordinary steel only when greater strength is required, or in a part that undergoes repeated shock, as the upper tumbler shaft, in which the stresses are repeatedly reversed.

The tensile strength of nickel-steel shafting is 90,000 to 100,000 pounds per square inch against 60,000 to 65,000 pounds per square inch for ordianry steel, and the elastic limit is much greater, provided the nickel content is 3 to 4 per cent, which is the usual percentage for shafting. Nickel-steel castings usually carry a smaller percentage.

The tensile strength does not represent the proper basis for comparison between the two steels, as the ability of nickel steel to withstand shock is much greater than the increased tensile strength over that of ordinary steel would seem to indicate.

When an ordinary steel shafting on a dredge is found too small in diameter it is advisable to substitute nickel-steel shafting of the same size, thus obviating any change of bearings, etc., that would be necessary if shafting of a larger diameter were used.

The shafts of the upper tumbler, the lower tumbler, and the ladder hoist are usually made of forged nickel steel. The only casting made of nickel steel is the upper tumbler when the shaft is cast integral with it. Nickel alone does not affect the hardness of steel to any great degree.

NICKEL-CHROME STEEL.

Chromium increases the hardness and wearing qualities of steel to a marked degree. When it is used in connection with nickel both strength and hardness are obtained, giving steel very useful for dredges. It makes an ideal material for gears performing severe service, and is extensively used for bull wheels and pinions, ladder hoist gears, etc. The teeth do not break nor wear so readily. If manganese wearing plates cause excessive slip, nickel-chrome plates can be substituted to advantage.

The hardness of nickel chrome can be controlled to a nicety by adding the correct percentage of chromium, but usually just enough is added so that the castings are machinable. As it is not necessary to put nickel chrome through such a severe heat treatment, it is practicable to make castings of this material where manganese castings could be made only with much difficulty. Round lower tumblers, buckets, ladder rollers, screen tires, and rollers give much better service when cast of nickel chrome than when made of ordinary steel.

HIGH-CARBON STEEL.

The tensile strength and hardness of steel increase with an increase in carbon, but the ability to withstand shock decreases. In "mild" steel castings the carbon runs between 0.18 and 0.25 per cent. When the carbon rises to 0.80 per cent, machining of the casting becomes difficult. For upper-tumbler castings, which are protected with manganese on nickel-chrome wearing plates, high-carbon steel is well suited, as there is slight chance of breakage and the faces hold up well under the plates. Most equipment for which high-carbon steel is used would have a longer life if made of nickel-chrome steel, but owing to the additional cost of nickel-chrome steel it is not always the more economical.

ELECTRICAL EQUIPMENT ON DREDGES.[a]

Electricity is used almost universally as motive power on gold dredges, except in isolated districts where electric current is not available and where the installation of hydroelectric or steam generating plants would be neither feasible nor economical.

Although many changes have been necessary in the type and arrangement of electric apparatus since electricity was first tried on dredges, the use of electricity has from the first proven its superiority to steam drive as regards cost of power, maintenance charge, and ease of control. The modern dredge is now supplied with heavy-duty motors that give efficient service and require minimum attention.

BUCKET-DRIVE MOTOR.

The design of the motor for driving the bucket line, for which power requirements vary through wide ranges and are at times excessive, is a real problem. When the size of buckets, the speed per minute, and the character of the ground to be handled are known, the motor requirements can be estimated closely for operations under ordinary conditions. However, should a very large bowlder or a reef of hard material be encountered the demand on the motor is immediately increased, and unless protective devices are arranged the motor will either stall or break some part of the bucket line or driving gear, depending on the torque that the motor is capable of developing and the strains that the buckets and gearing are capable of withstanding. Although it is hardly probable that the operator would allow the motor to remain stalled with full current on for a sufficient time to burn it out, should difficult digging of one sort or another—sticky clay, for instance—be encountered, the motor might be periodically so overloaded that the resulting high temperature would eventually break down the insulation and the motor would burn out. Burnouts have also occurred from the unsoldering of connections in the rotors by reason of excessive current caused by an overload.

The maximum reading of the power consumed on the main drive motors on the Yuba No. 14 dredge, while digging and raising ladder, was 652 horsepower; with the digging friction out, the maximum was 428 horsepower; and with buckets running empty 217 horsepower was required.

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[a] See also pp. 94, 101.

In addition to these requirements the bucket-drive motor must be capable of running the bucket line reversed, and also of slowly revolving the empty line for inspection and repairs.

These and other problems have been satisfactorily solved by electrical engineers, so that, generally speaking, by following standard specifications adopted by the foremost dredge manufacturers, anyone, although unfamiliar with the conditions surrounding electrical equipment on gold dredges, would probably get satisfactory results.

Motor economy is of importance, and motor efficiency is vital. Above all things it is necessary to keep the dredge running; a dredge is designed primarily as a money-making machine, and in order to get the best results there should be as little operating delay as possible. A dredge idle is making no money. It may be said that the loss of time incident to motor difficulties is not excessive. In extreme instances, or with new installations, it may run as high as several per cent of the total time. Motor trouble at one of the California dredging companies in 1914 caused only 26 hours' loss of time, or 1 per cent of the whole delay.

POWER CONSUMPTION BY ELECTRICAL EQUIPMENT.

The power consumed by the digging motor is usually 35 to 50 per cent of the total power consumption, and that of pump motors about 25 per cent, except when monitors are employed, when it may run as high as 40 per cent. Centrifugal pumps are used on account of the low head required and the large volume of water to be handled, and these are generally driven by direct-connected motors. The drive of the digging ladder is either directly by gear-connected motors or is by belt. Both of these drives work satisfactorily and each has its champions. Those favoring the direct gear believe that it shows lower operating and initial costs. In regard to the power consumed by these different methods comparisons are difficult, as power consumption on dredges varies greatly on the same dredge over different periods, according to the operating conditions encountered. Comparisons of working costs and power consumption of the Conrey dredges using the direct gear drive and some California dredges using the belt drive are shown by the following tables. It is difficult to make satisfactory comparisions between total working costs of dredges operating in different fields, and even more so to make comparisons of power consumption of the bucket drive, as there is where the greatest variance occurs. The following table[a] is interesting in its comparison of the amount of power consumed by the different Natoma dredges during one month's operation. As the operating time is fairly even, a study of the yardage in comparison with the power consumed illustrates the difference of the physical condition of the gravel worked.

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[a] Von Bernewitz, M. W., Dredging at Natoma, Cal.: Min. and Sci. Press, vol. 107, Dec. 27, 1913, p. 1017.

Power consumed by Natoma dredges during one month's operation.

Item.	Dredge No.									
	1	2	3	4	5	6	7	8	9	10
Bucket capacity, cubic feet	13.5	8.5	8.5	13.5	9	9	9	15	15	15
Height of bank being dredged:										
Above water, feet	4	3	6	2	12	6	20	10	3	16
Below water, feet	16	23	20	18	44	54	42	53	58	21
Daily operating time, hours	19	20	19	21	21	21	20	19	20.5	19
Gravel dredged, cubic yards	232,014	232,644	124,440	205,274	159,294	150,533	170,690	142,529	238,004	253,376
Power consumed, kilowatts	221,440	143,520	137,600	192,600	238,800	219,600	272,000	346,800	362,400	282,000
Power consumed per cubic yard, kilowatts	0.95	0.62	1.10	0.93	1.49	1.45	1.59	2.4	1.52	1.11

The power used by the No. 2 dredge was only 0.62 kilowatt as compared with 2.4 kilowatts a cubic yard for the No. 8 dredge. The total operating costs for these dredges over longer periods are given in a subsequent discussion of working costs. The following table, covering power consumption during an extended period, is furnished by Mr. Charles Kammerer, of the Conrey Placer Mining Co.

Power consumption of Conrey dredges from time of starting to July 31, 1915.

Item.	Dredge No.			
	1	2	3	4
Bucket capacity, cubic feet	7½	7½	9½	16
Total time operating, hours	44,950	41,754	50,867	23,520
Average depth, feet	22.1	28.1	40.8	45.3
Cubic yards dredged	7,890,000	6,268,000	8,646,000	10,025,000
Power consumed, kilowatt -hours per yard	1.55	1.89	1.73	1.36
Power cost per cubic yard, cents	1.27	1.55	1.48	1.09

Although dredges 1 and 2 were identical in size and equipment, it is evident that the operating conditions were different, for the power consumption on the one was 1.55 kilowatt-hours a cubic yard as compared with 1.89 kilowatt-hours on the other. The working conditions of the No. 3 and the No. 4 might be more nearly compared, but these dredges were different both in size and type, so that significant comparisons are again impossible, the No. 3 dredge being a flume dredge and the No. 4 a bucket-elevator stacker dredge with the standard type of tables and with the most powerful digging motor in use on a gold dredge.

The power consumed on the Dawson dredges of the Yukon Gold Co. was as follows during a three-year period, approximating closely 1 kilowatt-hour a cubic yard dredged:

Power consumption of Dawson dredges during three-year period.

Year.	Total power consumption.	Total yardage.	Power consumption a cubic yard.
	Kilowatt-hours.		*Kilowatt-hours.*
1913......	4,971,114	5,133,575	0.906
1914......	4,973,153	4,800,781	1.03
1915......	5,264,273	5,041,075	1.04

The varying amount of power consumed on the same dredge when digging ground of different textures is shown by the following table prepared from figures furnished by Mr. W. H. James, of Oroville, Cal.

Power consumed on a 6-foot dredge operating in the Oroville district, showing that difference in formation regulates power used.

MONTHLY FIGURES.

Cubic yards worked.	Power consumed.	Power consumed a cubic yard.	Nature of formation.
	Kilowatt-hours.	*Kilowatt-hours.*	
101,564	89,880	0.885	Sand filling.
91,822	77,040	.839	Do.
85,314	67,320	.789	Do.
68,577	86,640	1.263	Red soil and clay filling.
65,768	82,560	1.255	Do.
62,638	85,440	1.364	Do.

YEARLY FIGURES.

874,845	882,837	1.01	Sandy filling.
710,257	956,050	1.34	Soil and clay filling.

Most of the California dredges, and, in fact, a large part of all dredges operating to-day have the bucket ladder belt-driven from a powerful motor on the main deck, and this method has proved satisfactory. Although it was found necessary after a smaller motor had failed to do the work satisfactorily to equip the bucket drive of the Conrey dredge with a 550-horsepower motor capable of 700 horsepower for continuous periods, there is no dredge in California requiring such a large motor on the digging end, and the conditions of operation for some of the California dredges are more difficult than those of Conrey No. 4. The Yuba No. 14 dredge is equipped with a 400-horsepower motor, and the large Natomas dredges Nos. 8, 9, and 10, which have been subjected to perhaps the hardest digging of any gold dredges in the world, have practically the same equipment. The power consumption is estimated on those dredges at 89 kilowatt-hours a cubic yard.

The average power consumption a cubic yard on the No. 14 dredge for a period of nearly two years was 1.45 kilowatt-hours. The 18-foot boat of the Boston & Idaho dredge with easy digging ground requires a belt-driven motor or main drive of only 300 horsepower, showing how the motor requirement varies under difficult digging conditions. The dredge was first designed with 14-foot buckets; a new line afterwards added had 18-foot buckets, but no change in motor drive was required.

The Yuba No. 15 dredge (Pl. LIII, *A*), designed to dig 82 feet below water level and equipped with 17-foot buckets, will have a 500-horsepower digging motor, which is 100 horsepower larger than that of the Yuba No. 14 dredge.

RECENT TYPE OF DIRECT-CONNECTED MOTOR DRIVE.

The following description of the bucket ladder drive at Ruby, Mont., illustrates the most recent practice of the direct gear-connected motor drive.

The most powerful digging motor in use for gold dredging is on the Conrey No. 4 dredge. This motor is of 550 horsepower, with a 100 per cent overload factor, and especially designed ventilation. The control arrangement is a radical departure from cast-iron grids, and consists of a water rheostat. The device as installed at Ruby differs from the Westinghouse water-rheostat control mainly in the manner of cooling of the electrolyte. In the latter type, coils of pipe are placed in the large electrolyte tank and cooling water is pumped through them, but in the Conrey installation the cooling coils are in the dredge pond, being fastened to the side of the dredge. Pipes connect the coils to the electrolyte tank, and a small pump constantly circulates the electrolyte through the coils. This arrangement insures greater cooling surface; also, there is no chance for the coils to become choked by the electrolyte, as occurred when this method was tried in another dredging field, where the water pumped from the dredge pond through the coils was muddy.

Previous to the construction of the Conrey No. 3 dredge, the bucket drive on the Ruby dredges was by sprocket chains, whereas California dredges then, as is now the general practice in that State, were driven by a rubber belt from motors placed on the main deck. When the No. 3 dredge was constructed, the direct connection of the digging motor to the bucket chain by cut-steel gearing was first used (Pl. LIV, *A*). To eliminate undue shocks and overloads on the motor, should the bucket chain run against large bowlders, or reefs in the bedrock, a large friction-disk clutch was put on the first countershaft from the motor, and a circuit breaker was placed on the power line of the digging motor. One of the essentials of perfect service from direct gearing is cut gears with perfect alignment. On the No. 3 such alignment was insured by building a large steel base to carry the digging motor and by using two counter shafts and an upper tumbler shaft. After seven years of continuous service, this direct drive is giving entire satisfaction. It has been used on the No. 4 dredge since the large motor was installed. Some of the large dredges constructed in other fields have adopted the Ruby practice of direct gear drive.

KIND OF ELECTRIC CURRENT USED.

The problem of control in large motors for digging service would be simpler if direct-current motors could be advantageously used. However, as the use of these motors is impracticable the control must be designed for alternating-current motors. One of the reasons is that alternating current is furnished by the power companies, as direct current can not be economically transmitted from distant generating plants. The use of direct-current motors, therefore, would require a motor-generator set to transform the current. The direct-current motors are larger, more complicated, and, besides being more expensive in first cost, require more skillful supervision. In this connection Mr. E. G. Thunen,[a] of Hammonton, Cal., makes the following statement:

> A comprehehensive discussion of the relative advantages and disadvantages of the several different direct-current systems as contrasted with the number of alternating-current systems and motors which are now available would be very long indeed. Probably the chief grounds for the discussion of the comparative advantages of direct-current and alternating-current power systems for dredging are the relative economies of power transmission and excellence of motor-performance characteristics, particularly as regards speed and torque. The high-voltage direct-current systems would certainly have good power-transmission economy, and the alternating-current series motor, brush-shifting polyphase series motor, and others can be built with speed-torque characteristics approaching those of the direct-current motors. Of all the motors, direct current or alternating current, the induction motor is the simplest, least liable to get out of order, and cheapest. Direct-current motors would be almost ideal for some of the drives on a dredge, especially the digging motor, if proper protective devices, both circuit breakers and slipping frictions, were employed.
>
> In the event of encountering rock reefs the importance of protective devices would be many times greater with direct-current bucket drive than with alternating-current drive, not only to protect the motor, but also for the protection of the bucket line and gearing. The reason for this is the very high torque which the direct-current motor is capable of developing at reduced speed, while the torque of the alternating-current motor is limited.
>
> Drum-type controllers are thought by many engineers to be sufficient for digging motors up to 200 horsepower. There are a number of digging motors of this size operating satisfactorily with drum controllers which have also been used on digging

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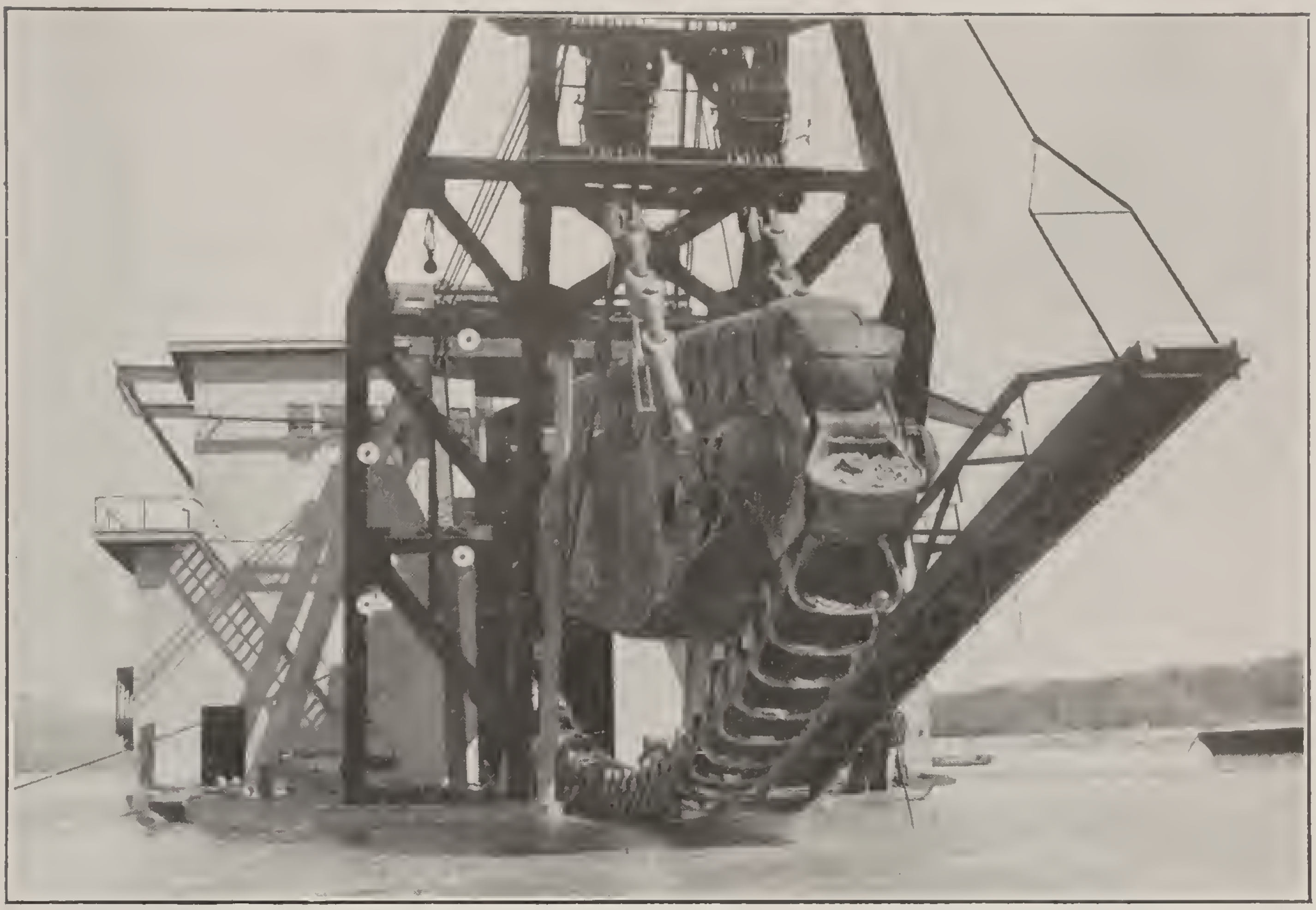
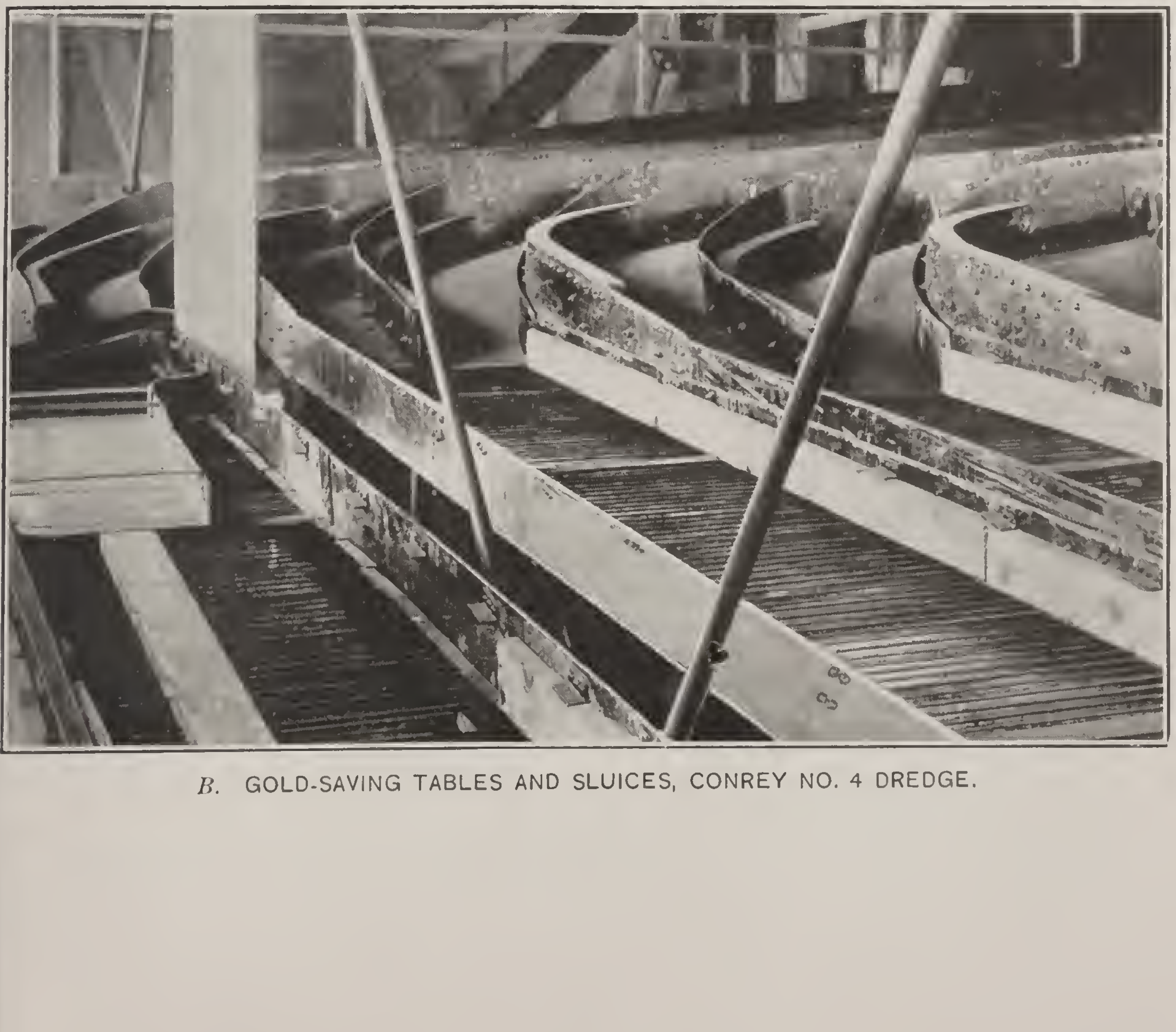
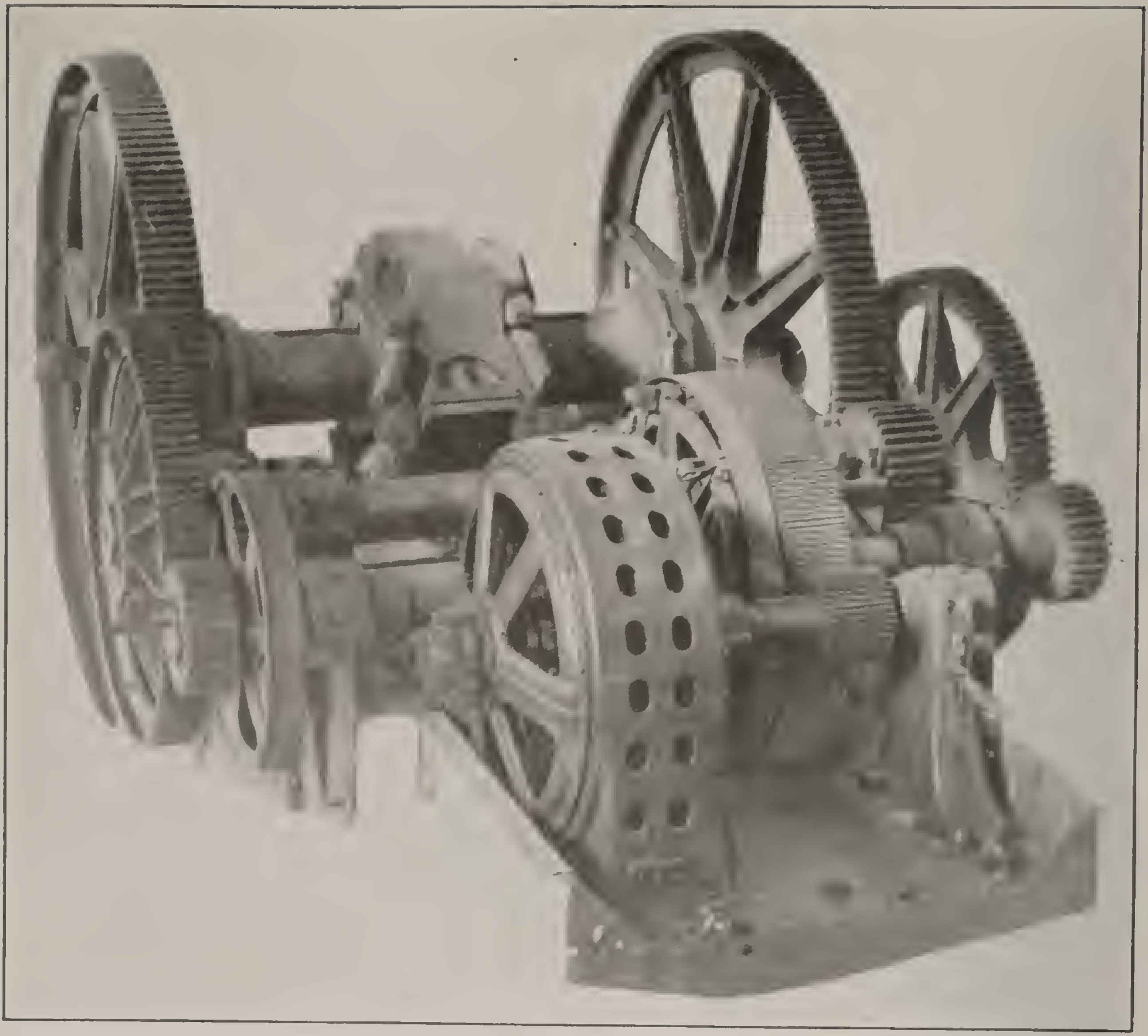
[a] Personal communication to author.



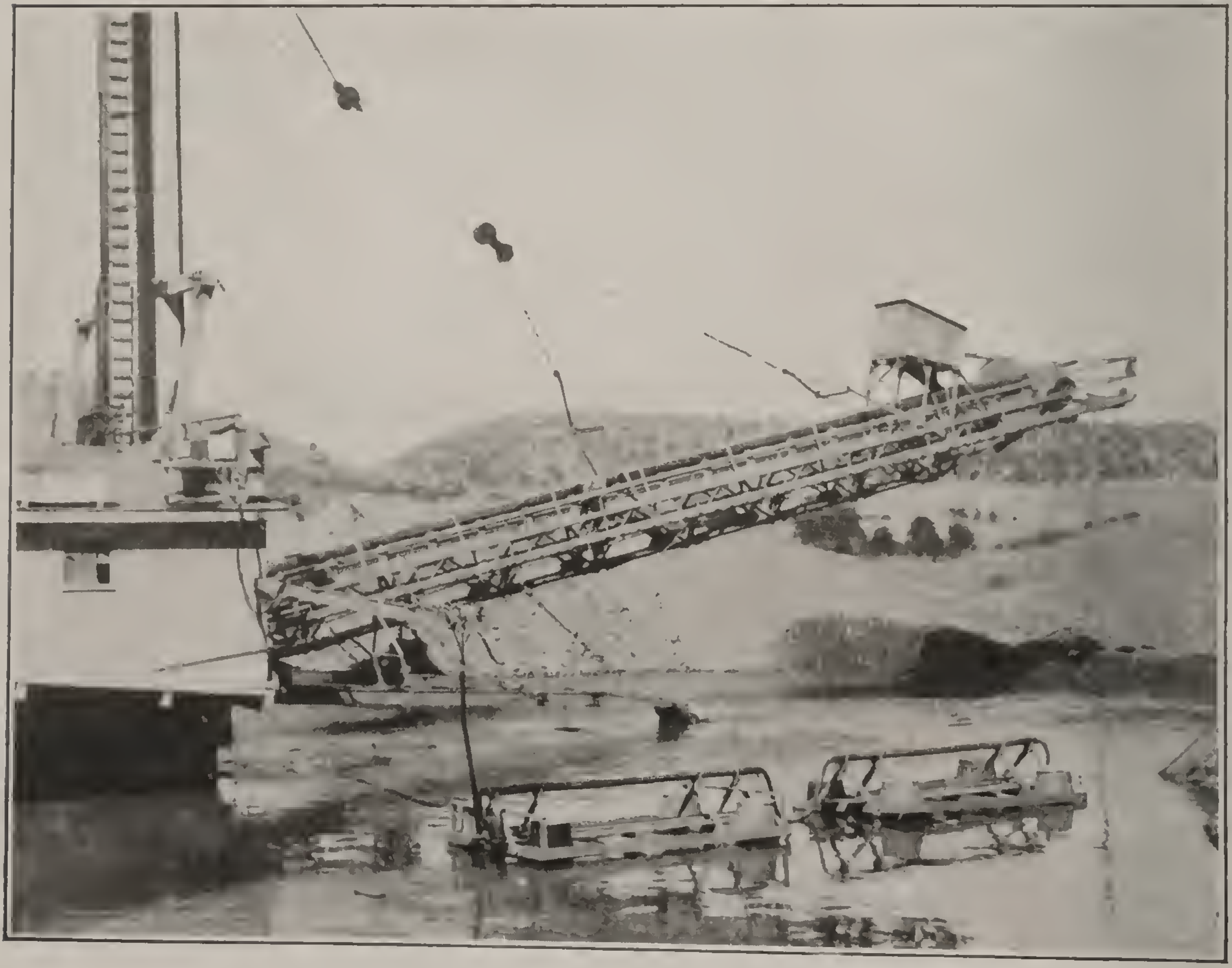
A. YUBA NO. 15 DREDGE, 17-FOOT BUCKETS.



B. GOLD-SAVING TABLES AND SLUICES, CONREY NO. 4 DREDGE.



A. DIGGING MOTOR OF CONREY NO. 3 DREDGE, WITH CUT-STEEL GEAR.



B. POWER-CABLE FLOATS, EL ORO DREDGE.

motors as large as 300 horsepower. For large motors, that is, those of over 200 horsepower, magnetic control or liquid rheostats are used, the latter being described above while the former are in general use in California; experiments made in that field with the liquid rheostat have not been satisfactory, perhaps for the reason mentioned above. Magnet-switch or grid control has been used satisfactorily on digging motors of the large California dredges operating under severe conditions, and no trouble has been experienced with same.

There is no doubt that a satisfactory combination of magnetic switch for the primary circuit of the motor, forward and reverse with overload relays, and of drum controller for cutting out secondary resistance, could be made for motors of 300 horsepower and over. This of course would do away with the automatic feature of uniform torque and current which, however, is not necessarily a disadvantage but is in fact a great advantage in certain cases.

The application of an ideal motor drive for the belt-conveyor tailing stacker is somewhat difficult. The requirement is severe for the reason that it is so often necessary to start with abnormal loads, and owing to the fact that the motor is generally placed at the far end of the stacker. The desire to run at reduced speeds under light load complicates the problem. A case in point is that of Yuba No. 14, where the stacker is 160 feet long; a high starting torque is required and it sometimes takes several minutes to bring the belt up to speed. Squirrel-cage motors with heavy-torque characteristics have been successfully used for this stacker drive, but some engineers prefer the slip-ring type, as a smaller motor can be used and they figure a consequent loss of power with the squirrel-cage type. The drive from the stacker-belt motor may be by silent chain belt, direct-gear, or ordinary belt. Each has its advocates and advantages. The stacker motor is small compared with the total motor equipment on a dredge, and a slight loss of efficiency is not as vital here as at the digging end. Revolving screens are motor-driven from either the upper or the lower end. Better practice seems to be from the lower end. On large dredges the screen motor must be designed to take a heavy starting load. When the screen is at rest and loaded with clay and rocks, as is often the case, considerable power is required to bring it up to speed. Some dredges are supplied with separate motors for ladder hoists. This is necessary when using a direct main-drive motor, and is also advantageous on other dredges, permitting greater flexibility of operation and consequent saving of time on account of being able to hoist or lower the ladder without interfering with the main-drive action. Electric current for dredge operation is usually transmitted from the main lines to a substation and from there transformed to the proper voltage for dredge use, from which it may be afterward stepped down on board the dredge. On the Yuba River the current is generally brought on board at 4,000 volts and stepped down to 440 for general use on the dredge. Most of the dredges of the Natomas Consolidated take the current aboard at 2,200 volts for use on all motors of 50 horsepower and over, whereas for smaller motors the current is stepped down to 440 volts as above.

VOLTAGE OF CURRENT USED ON DREDGES.

At Ruby, Mont., the current is transmitted at 46,000 volts to the substation, where it is stepped down to 2,200 volts, the voltage used aboard the dredges. Operating all motors at 2,200 volts makes it unnecessary to have transformers on board, and it is thought by some that, although the high voltage is more dangerous to workmen, there is less loss of power, and also less danger of fire, except on the all-steel dredge where the danger from fire is obviated. Using the higher voltage motors and dispensing with transformers reduces the weight of machinery on the hull. All motors on the large dredge of the Boston and Idaho Co. are designed for 2,200 volts. At Breckenridge the Tonopah Co. buys current at 13,000 volts, which is stepped down to 440 on all motors.

Thunen [a] states further—

> In selecting the electrical equipment for a dredge (or for any other purpose for that matter) it should be borne in mind that for the same economy of power transmission the cross section of line and power-cable conductors for the 2,000-volt system must be four times as great as that necessary for the 4,000-volt system. The other line construction materials—namely, insulators, poles, and cross arms—will be the same for both voltages.

Electric power is brought on board a dredge by means of armored insulated cables. Some operators use floats or small pontoons (Pl. LIV, *B*) to support the cable across the pond; others consider it better practice to carry the cable under water, because the insulation, if the floats are not well arranged and the cable gets wet and dry alternately, rots quickly unless thoroughly protected by rubber hose. In some places, as at Idaho City, Idaho, the current is taken on board the front end of the dredge and no floats are required; but winter conditions there are severe upon the cable insulation. A power cable supported on properly constructed floats, or pontoons between the shore and the dredge, is vastly easier to move than if unsupported. This is particularly noticeable on a large dredge digging wider than a single cut. The power cable for the large dredges weighs about 5 pounds a foot.

The table following shows the motor equipment on a number of different dredges:

Electrical motor equipment on dredges of Natomas Consolidated of California.

Name of dredge.	Size of buckets.	Average depth of cut.	Size of motors.				Pumps.		
			Bucket.	Swing.	Screen.	Stacker.	Group 1.	Group 2.	Group 3.
	Cu. ft.	*Feet.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*	*Hp.*
Natoma No. 1	15	32	300	35	75	25	150	35	30
Natoma No. 2	8	35	200	50	50	35	75	35	10
Natoma No. 3	8	40	200	25	50	35	75	35	30
Natoma No. 4	15	42	300	30	75	(2) 25	150	100	50
Natoma No. 5	9	50	200	30	50	40	100	25	50
Natoma No. 6	9	50	200	30	30	50	100	50	40
Natoma No. 7	9	58	300	30	50	50	100	50	30
Natoma No. 8	15	60	400	75	75	60	150	75	50
Natoma No. 9	15	60	400	75	75	60	150	75	50
Natoma No. 10	15	60	400	75	75	60	150	75	50

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[a] Thunen, E. G., Personal communication.

The following description by Gardner and Shepard [a] of the electrical equipment of the Yuba No. 14 dredge will be of interest, as it represents the latest practice in the California field:

The electrical equipment of the Yuba No. 14 is something of a departure from the equipment heretofore considered standard for California gold dredges. This departure from former practice was made in the light of experience gained in the operation of four dredges of the largest type, which were built by the Yuba Construction Co. for the Natomas Consolidated of California and one similar dredge built for the Yuba Consolidated Gold Fields. All electrical equipment was made by the General Electric Co.

Power is supplied by the Pacific Gas & Electric Co. as a three-phase, 60-cycle, 4,000-volt current. The power is brought on board the boat through 750 feet of No. 0 B. & S. gage, three-conductor cable, each conductor being stranded and insulated with $\frac{5}{32}$-inch 30 per cent Para rubber, conductors twisted together, rounded out with jute, and covered with a layer of $\frac{1}{8}$-inch varnished cambric. Over this is a layer of jute, and the cable is armored with No. 10 B. W. G. armor put on with short pitch. This cable is insulated for 4,500 volts working pressure. The shore cable is brought aboard the dredge on pontoons and enters a switch house on the upper deck near the stern of the dredge, where there is installed a nonautomatic, 300-ampere, 7,500-volt oil switch with hand-operated remote control. The switch is mounted on pipe framework away from the panel carrying the operating lever. There are installed in this switch house two 4,400/110-volt, 200-watt potential transformers with fuses, and two 150-ampere current transformers. The secondaries of these transformers are connected to indicating instruments mounted in the winch room. From the switch house the current is fed through triple-conductor varnished cambric cable in conduit to the primary side of the main transformer. There is installed one 15-kilowatt, 4,000-volt primary, 230/115-volt secondary, oil-cooled transformer to supply lights. From the secondary side of the main transformers, various feeder circuits supply the different motors, triple-conductor varnished cambric being used in iron conduits.

The control panel for the main-drive or digging motor, and for the winch motor, is placed in the winch room, as is also the instrument panel. The instrument panel contains the following instruments, which are supplied, as mentioned above, from the instrument transformers placed in the entrance switch house: One 5-ampere alternating-current ammeter with 150-ampere scale, one 175-volt alternating-current voltmeter with 8-point potential receptacle and plug, one alternating-current polyphase indicating wattmeter with 1,200-kilowatt scale. The readings of these instruments give the total input to the dredge.

The double-circuit control panel for the main-drive and winch motors has the following equipment: One 60-ampere alternating-current ammeter (winch motor); one 5-ampere alternating-current ammeter, 800-ampere scale (main-drive mortor); one 200-ampere automatic oil switch with double-coil series trip (winch motor); one 600-volt, 800-ampere automatic oil switch with double-coil trip (main-drive motor); and two 800-ampere current transformers in main-drive motor circuit.

The panels for the pump motors are placed on the lower deck, the starting compensators for the pump motors being mounted alongside the panels. These compensators are all provided with inverse time-limit relays. The pump-motor panels consist of the following: Two 3-phase 2-circuit motor panels, each panel mounting two 500-volt, 200-ampere lever switches controlling the high pressure, the low pressure, the 6-inch two-stage, and the vertical pumps.

The stacker and screen motors are controlled from independent panels in the stern of the dredge, each panel having the following equipment: One 200-ampere automatic oil switch, with 2-coil series trip, and a mounting on the front panel for one reversible type controller. The oil-switch operating lever is directly above the controller. These motors are of the slip-ring, variable-speed type, provided with resistance for continuous operation at 50 per cent to full speed, and with reversible controllers. They operate at 600 revolutions per minute, the screen motor being 75 horsepower and the stacker motor 60 horsepower. In most of the former gold dredges these motors were of the constant-speed squirrel-cage type. However, in the more recent large dredges, owing to difficulties sometimes experienced with squirrel-cage motors on account of the heavy starting conditions, and the desirability at times of running for short periods at reduced speeds, slip-ring motors, with resistances for 2-minute starting duty, were used. In heavy work, however, these were found too light, and for Yuba No. 14, resistances for continuous operation were installed and reversible controllers used. There is also provided, for the repair shop on board the dredge, a 2-horsepower, 180-revolutions per-minute, three-phase motor with pulley and base.

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[a] Gardner, W. H., and Shepard, W. M., The largest electrically operated gold dredge: Min. and Sci., Press, vol. 108, June 27, 1914, pp. 1055–1056.

All of the panels are natural black slate mounted on pipe supports.

The digging or main drive of the dredge is a 400-horsepower, 514-revolution-per-minute, 3-phase, 60-cycle, 440-volt, slip-ring, variable-speed, induction motor, with three bearings, pulley, and sliding rails, with a master controller and contactor panels, and is provided with resistance good for continuous operation at all speeds, from 50 per cent normal to normal.

The contactor equipment is provided with current-limiting relays, which limit the maximum load that the motor can take, and so protect both the motor itself, and the complete digging mechanism that it drives. This is of considerable importance in keeping down the cost of repairs, and preventing loss of time incident to repairs. The digging mechanism is subject at times to sudden and excessive loads, and without the current-limiting feature provided by the contactor control, the only protection would be given by the overload oil switch. Owing to the annoyance of having this switch tripping out frequently, it is usually set so high as to afford little protection. The current-limiting relays also protect the motor and digging mechanism when starting. The winchman can throw the master controller to the full "on" position, and the motor will come up to speed at a predetermined rate, which will not impose excessive stress on any part of the mechanism.

Another motor operates the winch, raises the spuds, and swings the boat. This motor is a 35-horsepower, 600-revolution-per-minute, 3-phase, 60-cycle, slip-ring, variable-speed induction motor, with a controller and resistance for continuous operation at 50 per cent to full speed. The motor is provided with pulley and base. In a few instances on smaller dredges, motors of intermittent rating were furnished for operating the winch. These motors, however, have not been found suitable for this service, as the winch operates almost continuously.

All the pump motors are squirrel-cage motors, the high-pressure pump having a 150-horsepower, 600-revolution-per-minute motor, the low-pressure pump, a 75-horsepower, 600-revolution-per-minute motor, the 6-inch two-stage pump, a 50-horsepower, 1,200-revolution-per-minute motor, and the vertical pump a 10-horsepower vertical motor. These motors are all direct connected to the pumps, and with the exception of the 10-horsepower motor, are all provided with welded end-ring construction in the rotor.

Practically the same conditions governing the electrical equipment on No. 14 obtain on No. 15 dredge, except that the latter dredge is designed for deeper digging. In addition to the larger (500-horsepower) digging motor on the No. 15 dredge, there will be a separate motor drive for raising and lowering the stacker, an additional hold pump (two hold pumps on No. 15), direct-current control for the main-drive contactor panel and switch arrangements to start the pump motors from the middle point of the main transformers in place of using compensators. The transformers will also be a little larger than those on No. 14.

Plates LV[a] and LVI[b] and figures 11[c], 12, 13, and 14[d], which include views of different dredges using electrical equipment show the placing of the motors.

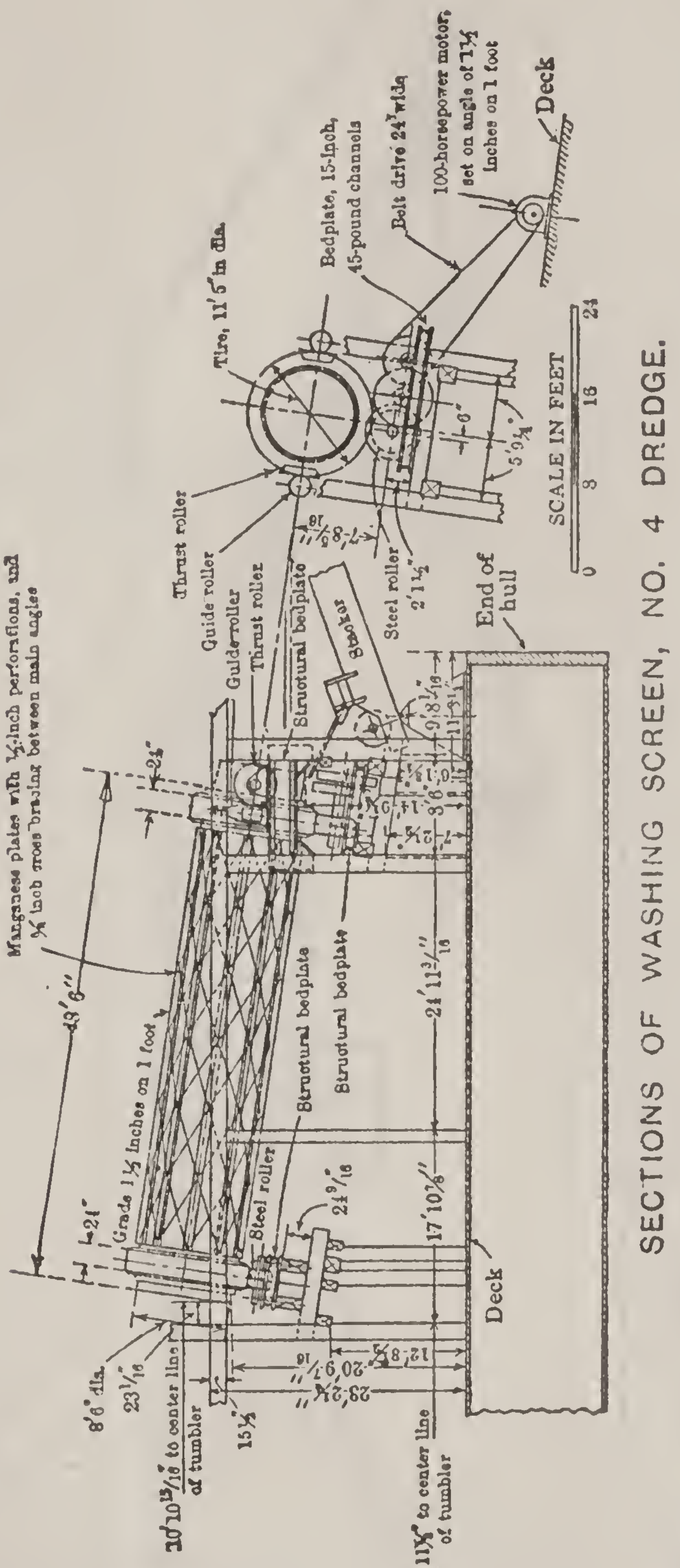

FIGURE 11.—Sections of washing screen of Conrey No. 4 dredge, showing placing of 100-horsepower driving motor.

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[a] Jennings, Hennen, The history and development of gold dredging in Montana: Bull. 121, Bureau of Mines, 1916, p. 18.

[b] Jennings, Hennen, work quoted, p. 16.

[c] Jennings, Hennen, work quoted, p. 18.

[d] Janin, Charles, Review of gold-dredging in 1912: Min. and Sci. Press, vol. 106, Jan. 4, 1913, p. 47.

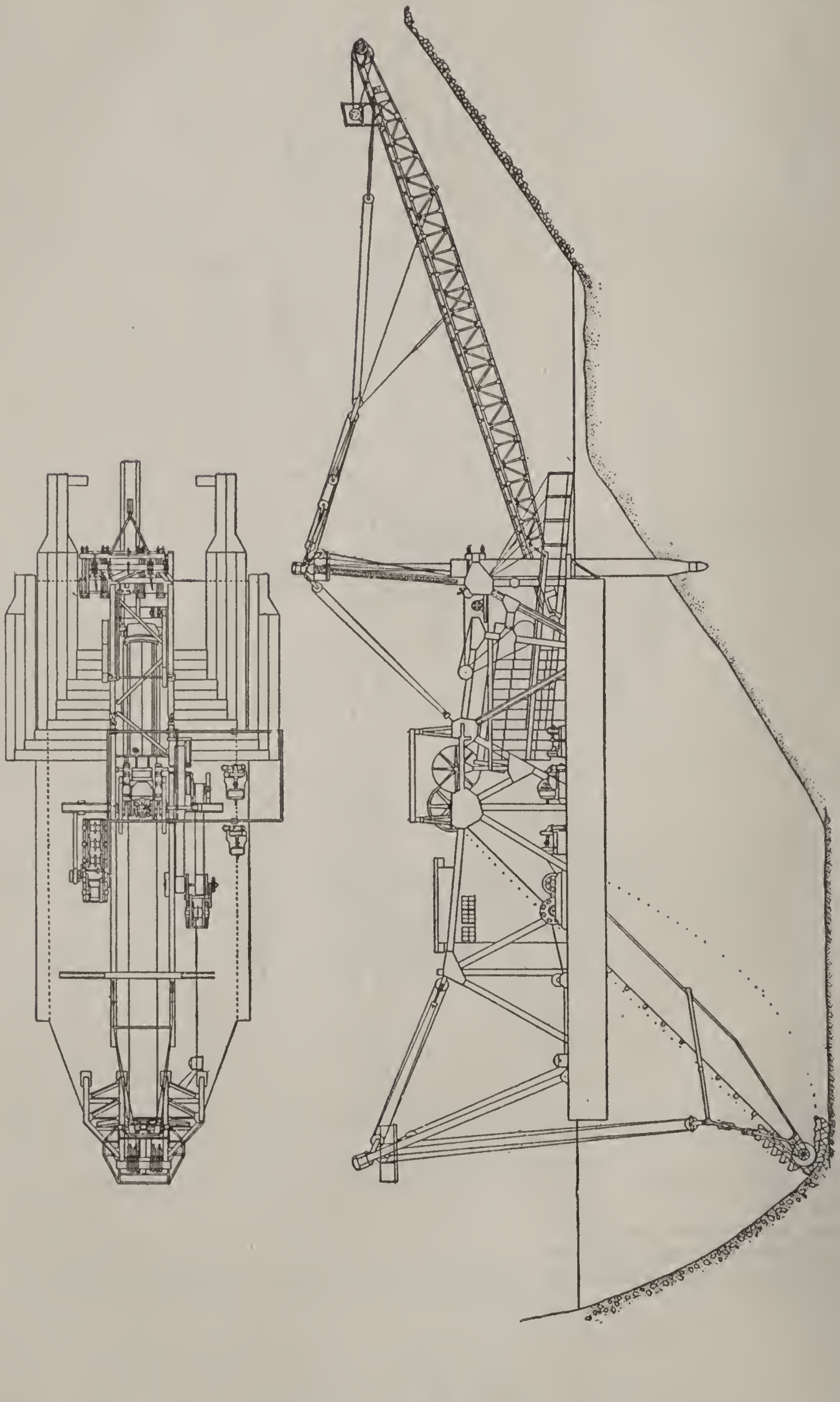
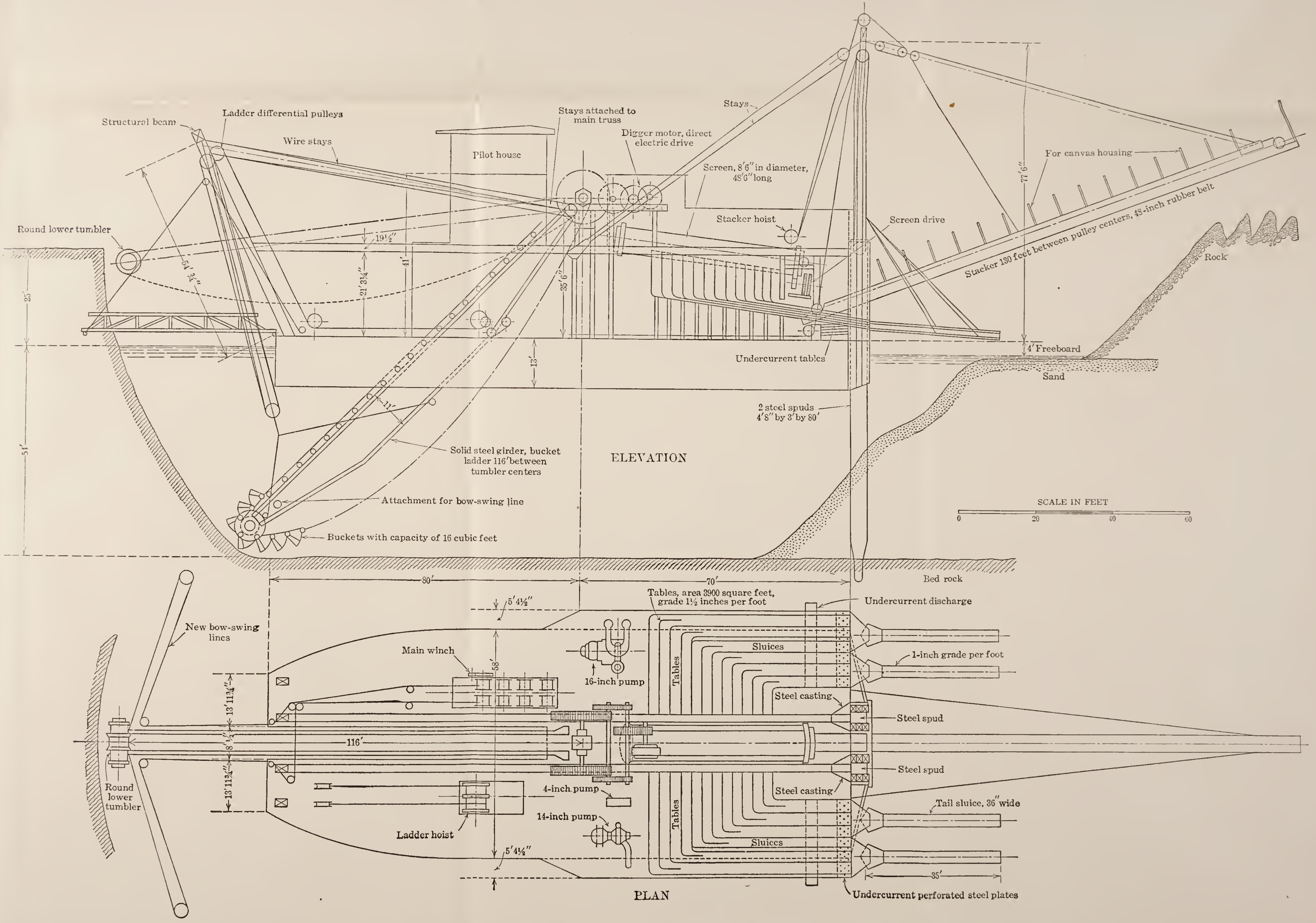
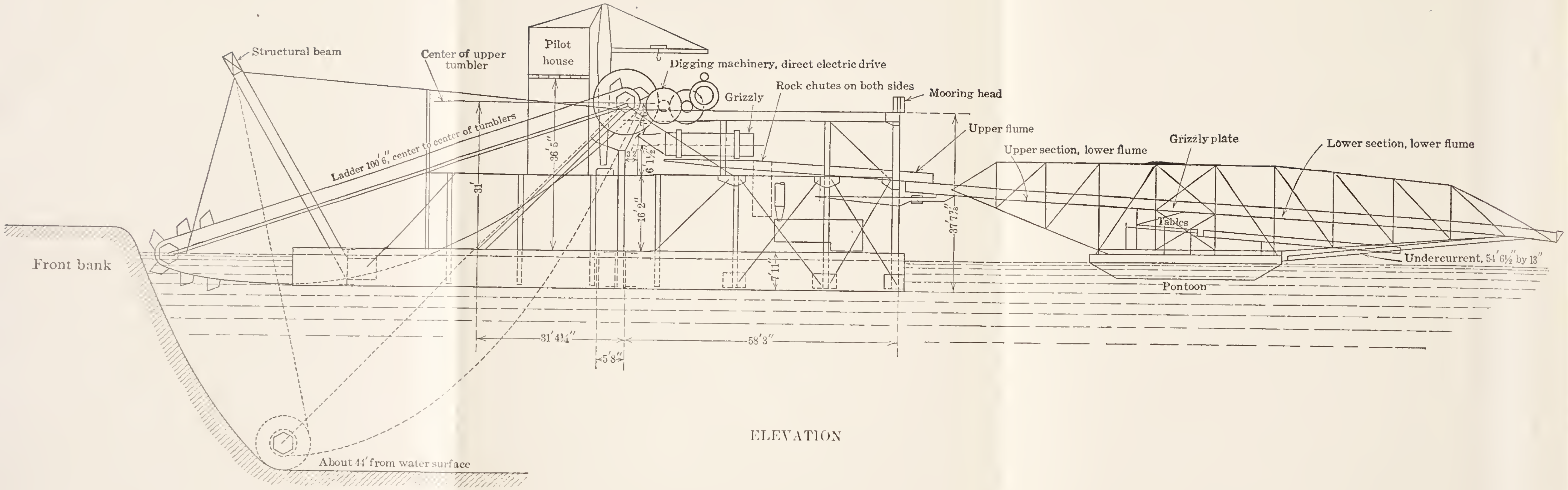


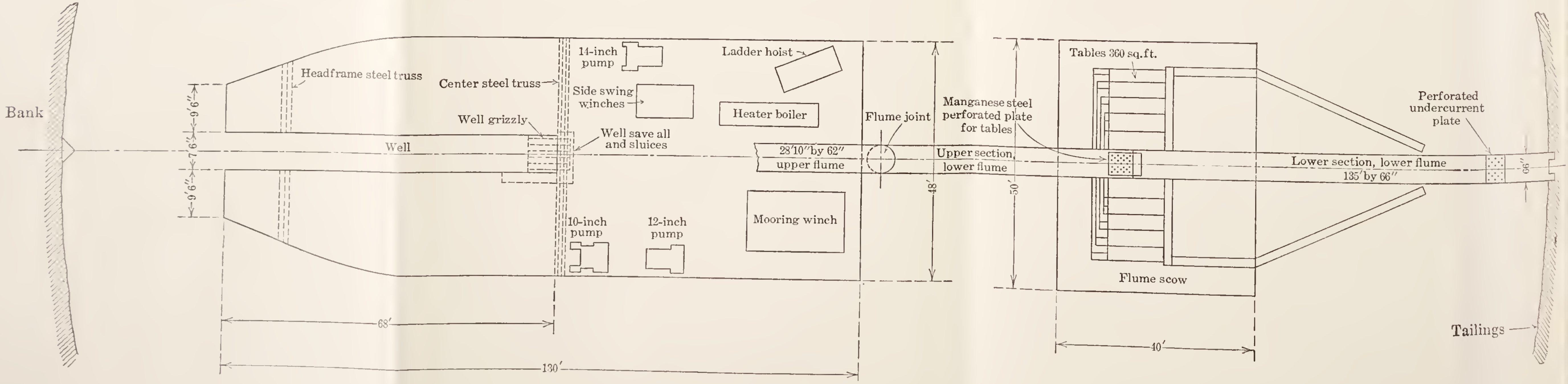
FIGURE 12.—Plan and elevation of electrically driven gold dredge.



SECTIONS OF CONREY NO. 4 DREDGE (MAIN BUT INCOMPLETE DETAILS).


Structural beam
Center of upper tumbler
Pilot house
Digging machinery, direct electric drive
Rock chutes on both sides
Grizzly
Mooring head
Upper flume
Upper section, lower flume
Grizzly plate
Lower section, lower flume
Ladder 100′ 6″, center to center of tumblers
36′ 5″
31′
3′2″
6′ 1½″
16′ 2″
37′ 7⅞″
7′ 11″
Tables
Undercurrent, 54′ 6½″ by 13″
Pontoon
Front bank
31′ 4¼″
58′ 3″
5′ 8″
About 44′ from water surface

ELEVATION



PLAN

SKETCH OF CONREY NO. 3 SLUICE DREDGE.

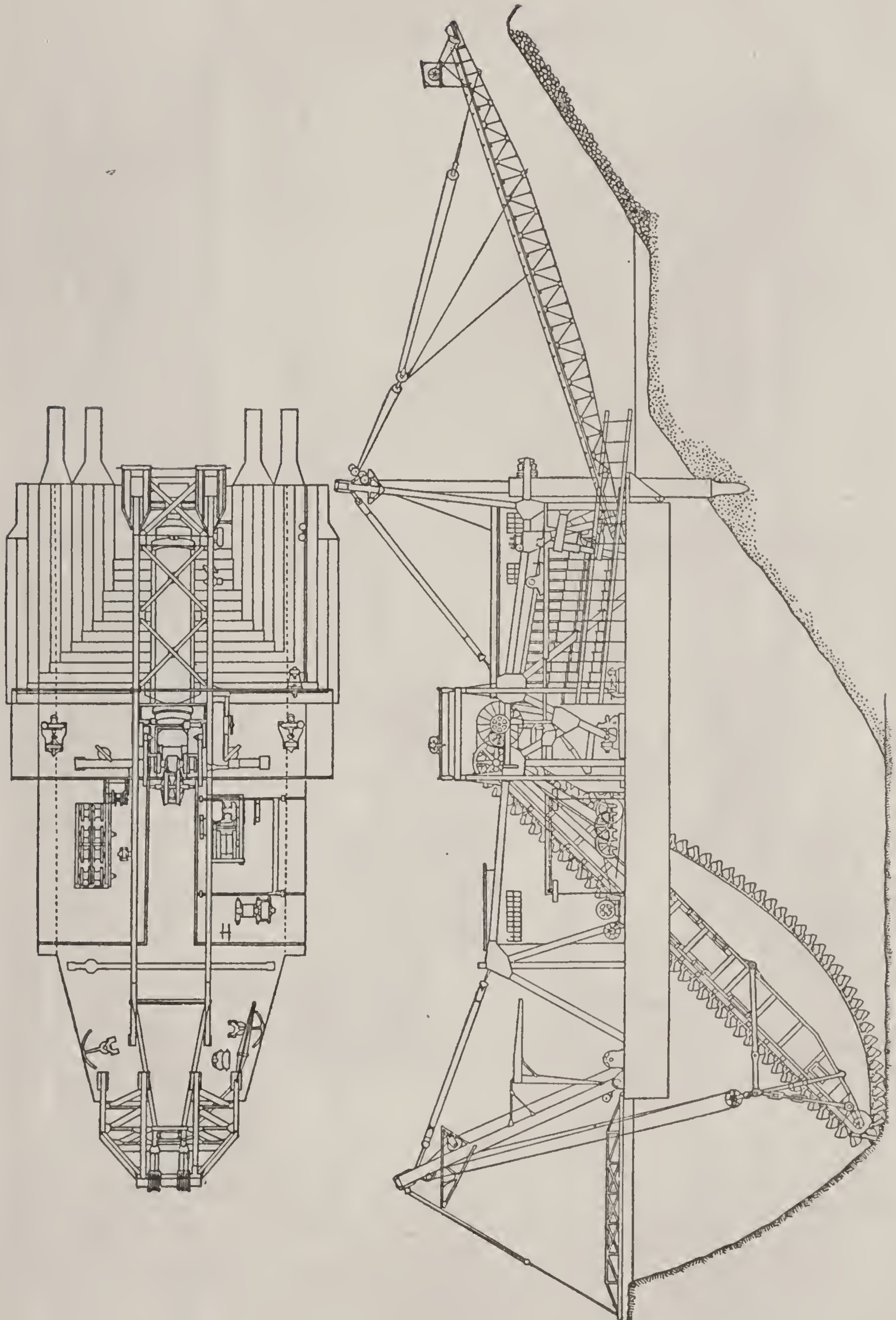


FIGURE 13.—Plan and elevation of 17 cubic foot Yuba dredge electrically operated, digging 80 feet below water level.

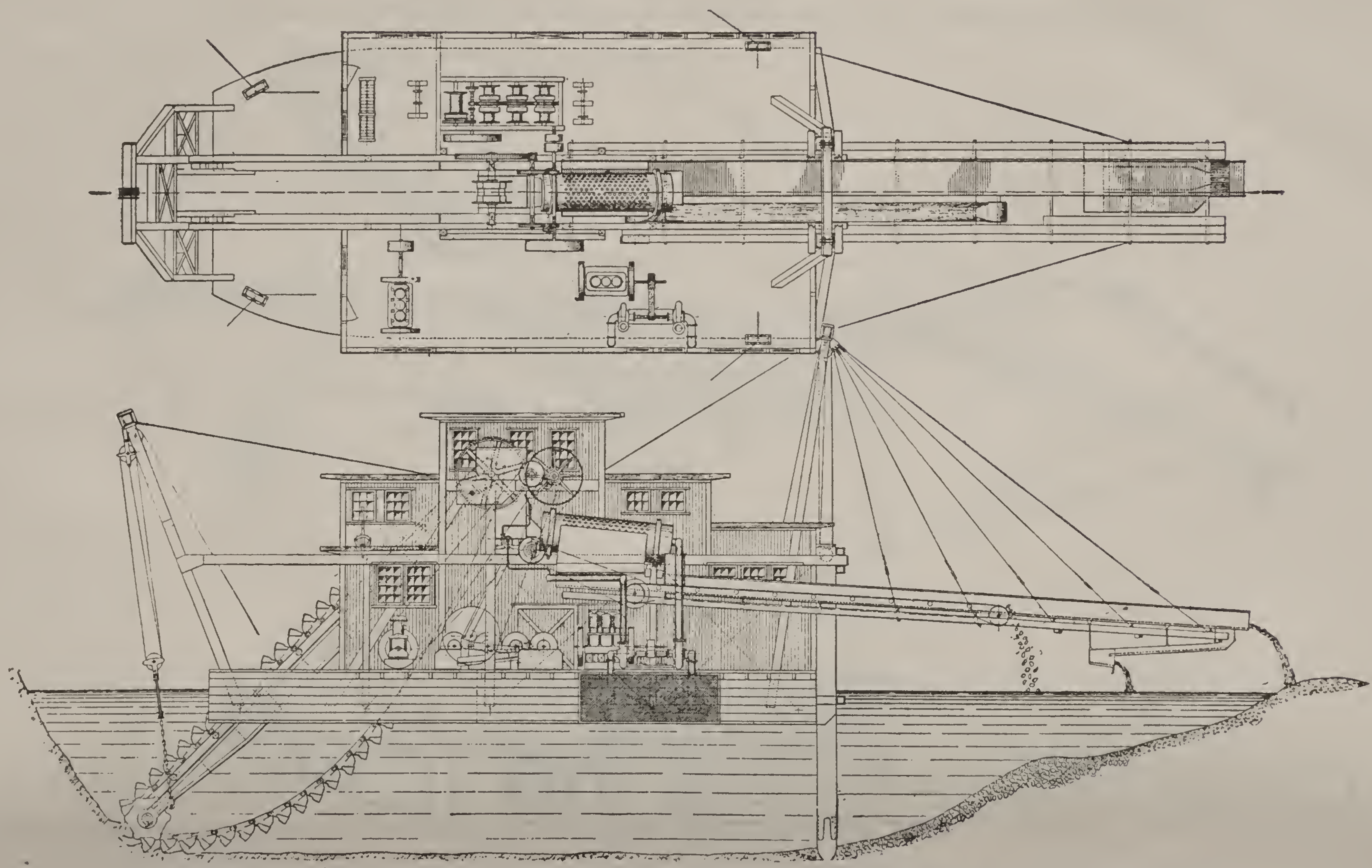


FIGURE 14.—Plan and elevation of flume-type Alaskan dredge with revolving screen and with distillate engines.

REBUILT DREDGES.

FEASIBILITY OF MOVING DREDGES OR MACHINERY.

A matter of considerable interest at present is the moving of machinery from dredges that have worked out the areas for which they were built or have been dismantled and replaced by other dredges. Areas that a few years ago were deemed too small for profitable dredging are being considered on the basis of modern practice and with the idea of reconstructing a used dredge. The machinery of some of these dredges that have been or are to be dismantled is in good condition and fit for many years of service; and on properties not too difficult of access it can be refitted to new hulls and practically new dredges built, in some instances at less than 50 per cent of the original cost. These rebuilt dredges may not always be adapted to handle the gravel at as low operating cost as might be attained with new dredges, but the smaller expense of installation will prove a large factor in their selection and use. The machinery of two dredges that proved unprofitable at Golden, Colo., was moved to new hulls, one dredge being rebuilt in the Breckenridge district in the same State and the other at Jenny Lind, Cal., and both dredges yielded a profit in their new locations.

EXAMPLES OF SECOND USE OF DREDGES OR MACHINERY.

Dredge machinery from dismantled hulls in California has been moved to Alaska, and a number of dredges have been dismantled and rebuilt on other areas in the same or in a different State. Several of these dredges were financial failures in the first locality and proved successful afterwards, as were the Golden dredges, but with one dredge at least poor judgment seemed to follow its career.

TIN CUP DREDGE.

This was the Tin Cup, a 3-foot dredge, originally built near Shoshone, Wyo., at a cost of approximately $49,000. The recovery from the ground for which it was built was less than 1 cent a yard. The dredge was afterwards purchased for about $11,000 and moved to Gunnison County, Colo., at a total cost stated to be approximately $65,000, or more than a similar dredge would have cost new. The second field of operations was seemingly no better chosen than the first, as the dredge was sold at a sheriff's sale in 1915.

SCOTT RIVER DREDGE.

Examples of a more favorable kind may be mentioned. The Scott River dredge, first operated at Callahan, Siskiyou County, Cal., was unprofitable, and it was shut down in May, 1910, after it had worked a little less than two years. This dredge, which was equipped with 7½ cubic foot buckets, was purchased by the Alta Bert Gold Dredging Co., and the machinery moved to Trinity County, Cal. The total cost of the rebuilt dredge was approximately $90,000, including purchase price of machinery, freight charges, cost of hauling 28 miles at 1 cent a pound, and cost of building a new modern hull and a power line 5 miles long. This dredge has been working successfully for several years.

BUTTE DREDGE.

The Butte dredge, which was equipped with 3½ cubic foot buckets, was built and put in operation at Oroville, Cal., in November, 1902, at a cost $52,000, and was dismantled in July, 1910, after handling nearly 3,250,000 cubic yards. The machinery, valued at $6,000, was moved to Jenny Lind, Calaveras County. The cost of the rebuilt dredge, including all charges, is given below. It will be noticed that the cost of new machinery, which consisted principally of new buckets, was $9,446.

Cost of rebuilt Butte dredge.

Estimating and traveling expenses		$237.23
Stationery, drafting, etc		905.80
Superintendence		1,910.00
Hull and superstructure:		
Lumber		4,655.61
Hull irons and spikes		1,005.41
Labor		4,650.95
Paints and materials		347.15
Hardware and sundries		1,212.73
Machinery installation:		
Repairs	$910.28	
Freight, hauling, etc	3,012.44	
Labor and sundry materials	5,033.00	
Electric wiring and equipment	1,014.32	
		9,969.94
Total cost of moving and rebuilding		25,356.52
Outside expense:		
Camp	$727.24	
Water plant	771.71	
Power line, transformers, etc	2,673.50	
Changes in construction	1,900.22	
		6,072.67
		9,446.40
Value assessed to old machinery		6,000.00
Total cost of rebuilding dredge		46,875.59

EL ORO NO. 1 DREDGE.

The machinery of the El Oro No. 1 dredge, a 5-foot dredge, was moved from Oroville, Cal., to a new location at Yreka, Cal., where the dredge was rebuilt, after being redesigned. The cost of moving the machinery and rebuilding the dredge was approximately $55,000, the original company carrying out the work of rebuilding and operation. Other companies operating dredges in California have moved machinery from their dismantled dredges and advantageously reconstructed the dredge in a new locality. One of these dredges, the Pacific No. 1, was moved from Oroville to new ground on the American River, the total time required from commencing to dismantle the dredge until the rebuilt dredge started operations being just four months, which was rapid work.

DREDGE NEAR LEADVILLE, COLO.

A rebuilt dredge which cost more than was first anticipated but which has been a success is that known as the Magpie, on the Derry Ranch, near Leadville, Colo., originally built in South Dakota at a cost of approximately $105,000 ready to start. The dredge was unsuccessful and ceased operations. It was purchased by the New York Engineering Co., which had arranged to dredge the Derry Ranch property on a royalty basis. Dismantling of the dredge required three weeks. It was moved to the Derry Ranch, where construction commenced May 20, 1915. The dredge began operations October 10, 1915. Many important changes were made in construction, the hull was lengthened about 10 feet, and adjustments were made in the machinery to meet these conditions. The total cost of building the new hull and reconstructing the entire dredge, including cost of machinery, which was purchased at a low figure, is stated to have been approximately $100,000.

CASTLE CREEK DREDGE.

The Castle Creek dredge, built in 1912 at Mystic, S. Dak., did not prove the success anticipated and was moved to John Day, Oreg., and rebuilt for the Empire Mining Co. The rebuilding of this dredge was done under contract by the Stearns Roger Manufacturing Co., of Denver, the original constructors of the dredge; and the total cost of the dredge rebuilt ready for operation at John Day was figured at considerably less than the cost of a new dredge. At the time of writing, complete details were not available, but it is of interest to note that dismantling the dredge machinery and the power house and loading the equipment on cars at Mystic cost $4,500. The framed timber of the front, center, and rear gantries, the main trusses, and the screen frame, amounting to nearly 45,000 feet, was taken, and more lumber would undoubtedly have been saved except for the high freight rate to the new locality.

FEATURES TO BE CONSIDERED IN REMOVAL OF DREDGING EQUIPMENT.

If the machinery is in good condition, the company owning it can afford to consider rebuilding the dredge in a new locality, as a substantial saving can be made as compared with the cost of a new dredge. Where it is necessary to purchase the machinery of an old dredge, except in exceptional circumstances and at a reasonable price, it is questionable how great a saving can be made.

Among the points other than the condition of the machinery that are to be considered but are liable to be overlooked in valuing an old dredge, are the costs of dismantling and hauling to the nearest railroad; also parts of the superstructure of some dredges can be used to good advantage if the material is in good condition and freight charges not too high. The hull of a dredge is of no value unless the dredge can be moved bodily—that is, can dig its way to a new location. This was done by the Pennsylvania dredge at Oroville, Cal.

OTHER EXAMPLES OF MOVING DREDGES.

The Pennsylvania, after the operating company's property had been worked out, was purchased by a group of Oroville dredge men for $7,500 and paid for by a stock interest in a new company they then formed. The dredge dug its way to the new ground through a distance of 1 mile, the greater part of which was dredge tailing, in three months. About five weeks was required to overhaul the dredge before operations were begun on the new property. This dredge, although an old machine, having been first put in operation in November, 1902, will with care last the life of the property, and is already making a substantial profit for the new owners.[a] The advantages in this case were exceptional; an area of 40 acres of known value as dredging ground that the owner had for years refused to sell was purchased on satisfactory terms by experienced dredge men, who also had the opportunity to buy the dredge just as the ground for which it had been built was worked out.

Dredges built in Alaska which proved unprofitable in the original localities have been moved to new areas. One of these, the Osborne Creek dredge, was first built in lower Dry Creek at Nome. The ground had not been prospected before the dredge was built and when the dredge was started the gravel was found to be of too low grade for profitable dredging. The dredge was afterwards moved to Osborne Creek, about 12 miles distant. The machinery was first removed and the hull sawed in half and hauled in two sections. Each section weighed 55 tons and was drawn by 66 horses.

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[a] Since this was written the dredge has worked out the property and has again been sold.

In 1912 the No. 7 dredge of the Yukon Gold Co. was moved at considerable expense from Dawson, where it had worked out the property for which it was built, to Flat Creek, where it has since operated successfully. Other dredges of the Yukon company are being moved to Idaho from Bonanza Creek in the Klondike.

The following tables show the approximate maximum values given to different used dredge parts in California; the first covers a 5-foot dredge and a 5½-foot dredge, and the second a 9-foot dredge in excellent condition, with a new manganese-steel bucket line. It is thought that the figures may be of assistance to persons desiring estimates under like conditions, but it should be remembered that the purchase of second-hand dredge machinery requires considerable study.

Approximate value of second-hand parts of two dredges used in California.

Name of part.	5-cubic-foot close-connected remodeled dredge built in 1911.		5½-cubic-foot close connected remodeled dredge.	
	Condition.	Approximate value.	Condition.	Approximate value.
	Per cent.	*Dollars.*	*Per cent.*	*Dollars.*
Buckets	Poor.			6,000
Pins				
Upper tumbler		300	New.	800
Lower tumbler	Poor.			
Ladder			Good.	2,000
Ladder end casings	O. K.	50		250
Ladder suspension shape	O. K.	50		75
Ladder rollers and bearings				300
Bull wheel	Fair.	150		500
Intermediate shafts and bearings		250		500
Pulley		125		
Hopper	(a)	100		200
Screens	25	250	25	500
Screen drive	50	200		250
Ladder hoist		500	Poor.	
Main winch		900		
Operating levers	50	200		200
Ladder suspension tackle		500		500
Stacker suspension tackle		100		150
Spuds	None.		Good.	750
Deck sheaves	50	100		100
Distribution and gold-saving tables	20	100		100
Save-all		100		75
Spud casing	None.			100
Tail sluices				
Stacker chute				
Riffles				
Stacker ladder	75	500	25	600
Stacker rollers	None.		None.	
Stacker gearing		50		75
Bow gantry cap		100		125
Stern gantry cap		75		100
Pumps and piping	50	400		500
Ropes and bolts	20	75		75
Electric motors		1,500		2,000
Electric wiring		50		100
Tools and sundries		300		400
Hull and housing		50		100
Total		7,175		17,425
Approximate weight of all parts listed ... tons	50		150	
Cost of hauling to railroad	[b] $5	[c] $250	[b] $4	[c] $600
Cost of dismantling		$750		$1,500

[a] Part of hopper in fair condition.
[b] Per ton.
[c] Total.

Approximate present value of machinery and equipment on a 9-foot dredge built to dig 55 feet below water level.

Name of part.	Weight.	Cost new.	Salvage value.	Present value.
	Pounds.		*Per cent.*	
Buckets, 2 pieces	[a] 79	[b] $17,380.00	75	$13,035.00
Buckets, 3 pieces	[a] 16	[c] 3,456.00		Scrap.
Bucket pins	[a] 95	[d] 2,470.00	25	617.50
Bucket bushings	[a] 190	[e] 566.20		Scrap.
Digging ladder	150,000	9,000.00	50	4,500.00
Ladder rollers, complete	[a] 15	[f] 1,320.00	25	330.00
Ladder suspension, bars and blocks	45,000	4,275.00	50	2,137.50
Lower tumbler and shaft		1,750.00		Scrap.
Upper-tumbler body		1,000.00		Scrap.
Upper-tumbler shaft		1,000.00	75	750.00
Main drive, complete	45,200	4,300.00	50	2,150.00
Bucket ladder, complete	10,500	1,325.00	50	662.50
L. H. winch, complete	42,000	5,040.00	60	3,024.00
S. L. winch, complete	40,000	4,800.00	60	2,880.00
Main hopper	15,000			Scrap.
Revolving screen, complete	47,700	2,860.00	50	1,430.00
Revolving-screen drive	12,600	1,300.00	50	650.00
Distributor	36,000	2,160.00	40	864.00
Stacker ladder	46,000	2,400.00	50	1,200.00
Stacker-hoist winch	3,200	400.00	60	240.00
Stacker suspension, complete	5,000	500.00	60	300.00
Stacker drive, complete	16,500	1,450.00	50	725.00
Stacker belt				Scrap.
1 spud [g]	62,000	3,100.00	50	1,550.00
1 spud suspension	4,500	450.00	60	270.00
Deck depression and fairlead sheaves	29,500	2,500.00	50	1,250.00
Operating levers	11,500	1,150.00	50	575.00
Motors		8,000.00	65	5,200.00
12-inch high and low pressure pump	11,000	1,300.00	50	650.00
5-inch hopper pump		500.00		Scrap.
Total				44,990.50

[a] Number.
[b] At $220 each.
[c] At $216 each.
[d] At $26 a pound.
[e] At $2.98 each.
[f] At $88 each.
[g] One spud was valueless, being cracked in several places.

OPERATION OF GOLD DREDGES.

The material following is intended to describe dredging operations in a general way, although elsewhere in this bulletin will be found material that could properly be classed under this head.

The principal factor in operating a gold dredge is to maintain the highest possible proportion of operating time; that is, to keep the dredge digging. As mentioned elsewhere, a dredge idle is making no money, but fixed and interest charges still go on. A few years ago an operating time of 75 per cent was considered a high average. The average operating time of the modern dredge has increased to 80 or 85 per cent, mainly because of improvements in machinery and design and general strengthening of dredge parts that are subjected to great strain, although the weight of the machinery and the first cost of the dredge have greatly increased. The modern gold dredge has increased in weight from the early boats with a total of 300 tons of machinery and hull to the complete 16-foot dredge which will dig 80 feet below the water level and which weighs 2,250 tons.

ESSENTIALS OF DREDGE OPERATION.

With a dredge suitably equipped and placed, the essentials of operation may be summarized as follows:

(1) Outline the course with due regard to elevation of surface.

(2) Secure side lines to as long head lines as practicable.

(3) Step to insure a full bucket only.

(4) Run buckets at full speed in free ground; run more slowly in hard ground.

(5) Swing fast for a shallow drop and run slowly with deeper drop. The proper drop can be determined only on the ground.

(6) Cleaning of bedrock depends on situation of pay ground and careful cleaning of bedrock, digging well into it, may or may not be advisable.

(7) Motors should be in charge of an experienced electrician. On small dredges conveniently situated, an electrician can come at regular intervals and on special calls. The regular oiler can look after the oiling of motors daily.

(8) Keep repair parts in liberal supply, and of the best material.

AMPLE SPARE PARTS SHOULD BE AVAILABLE.

Perry[a] points out that in order to make a success of dredging operations far from sources of supply, the equipment must be complete and of the best possible kind. A single breakdown or loss of a part that can not be readily replaced may mean the loss of the season's work, so that the supply of extra parts and the shop equipment for repairs should be much greater than is ordinarily required. An ample supply of spare parts is advisable in all gold dredging, as can be illustrated by a case in point. During 1915 and 1916 the Natomas Co. operated under contract the 9-foot dredge of the Wilkes-Barre Co. This dredge was similar in design to several Natomas dredges, and the Natomas Co., with its complete organization and shop equipment, could work to far better advantage than a company operating a single dredge could possibly have done. For instance, when there were breakdowns of the Wilkes-Barre dredge, the repair gang would be rushed to it as though it were a Natomas dredge; and when in the fall of 1915 the bucket line began to go to pieces all at once, the Natomas Co. was able to supply from its large stock on hand enough extra buckets to keep the dredge running until the new buckets arrived, whereas if a company with a single dredge had been operating, a shutdown of two months or more would have probably resulted while new buckets were awaited.

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[a] Perry, O. B., Development of dredging in the Yukon territory: Trans. Canadian Min. Inst., vol. 18, 1915, p. 30.

Most dredges built in accordance with California practice are now equipped with digging spuds, in place of the head lines used on the early dredges. In easily dug light gravel, as that worked by the 16-foot Boston & Idaho dredge at Idaho City, the dredge was operated on head lines, and it was not necessary to furnish spuds, though the dredge had been designed for spuds and spud casings had been supplied when the dredge was built. Head lines have sometimes replaced spuds successfully when the dredge hull has become too old to withstand the strain of digging against the spud; and during the repairing of a broken spud, digging has sometimes been continued by rigging up a temporary head line.

DIGGING ON SPUDS.

The digging face is arc shaped; sometimes a single cut is made and at other times double cuts are carried. In carrying the excavation forward, the dredge is swung from one side of the cut to the other by side lines. Figure 15 shows a method of dredging to leave a channel. The spuds are raised or lowered by wire cables running through large sheaves on the rear gantry to the spud drums on the main winch. The digging spud is generally heavier than the stepping spud and is made of steel. Formerly the stepping spud was made of wood. The practice on large modern dredges is to have both spuds of steel and of similar weight and design; thus should the digging spud break the dredge can work on the other. The advantage is apparent.

To move the dredge forward, the spud is raised and the line on one side of the dredge is hauled in while the other side line is played out, the second spud being used as a pivot on which the dredge is moved. The first spud is then dropped, the second one raised, and the operation repeated. With each swing the dredge is stepped ahead 2 feet or more, depending on the width between spuds. For the new cut the total distance of moving the dredge is 5 to 8 or 10 feet. On the Yuba dredges the spuds are placed about 14 feet apart and it is necessary to swing through an angle of only about 40° to make a 6-foot step. The lower tumbler is usually greased during this operation, which requires about 12 minutes.

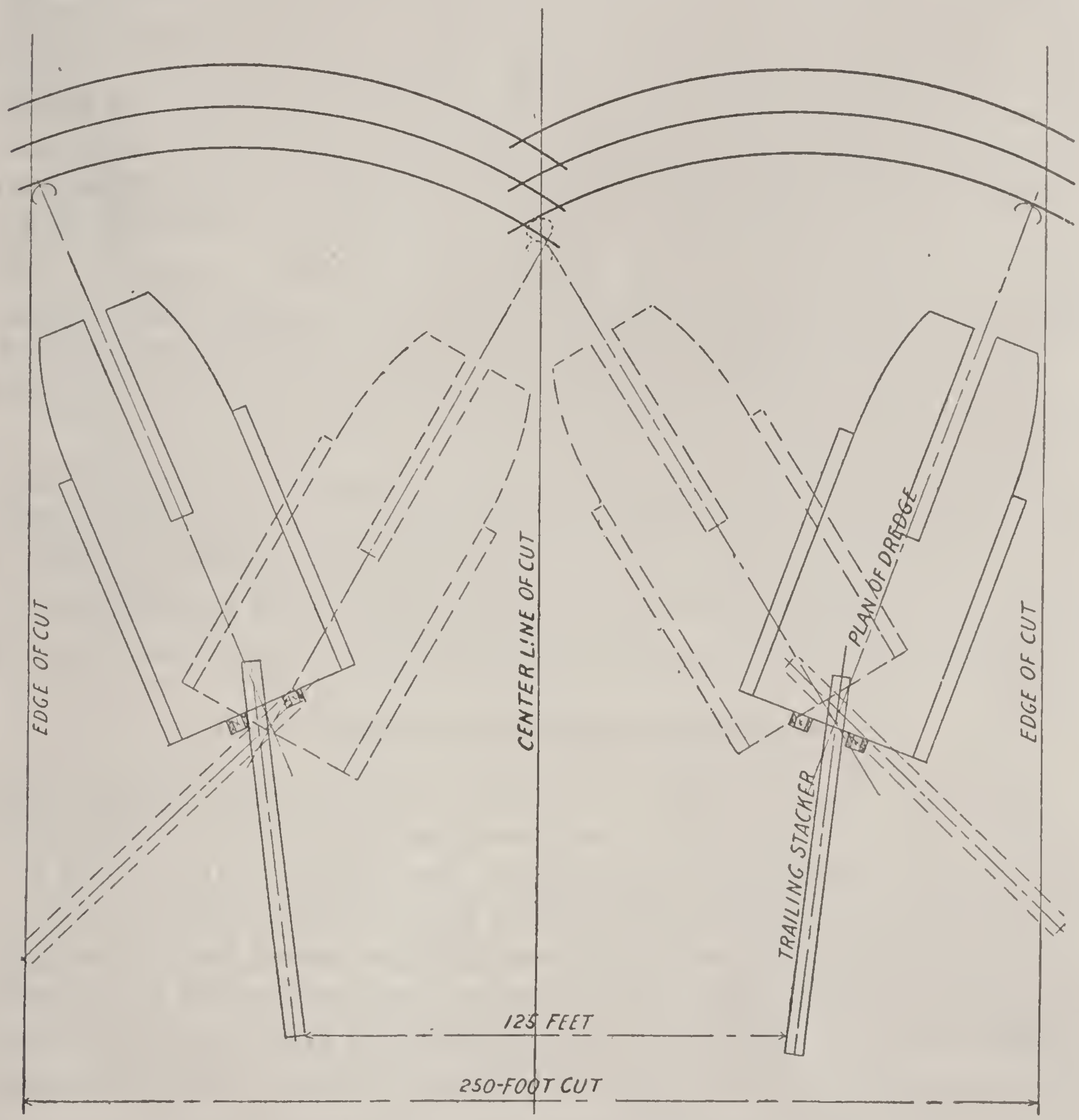

FIGURE 15.—Method of operating gold dredge to leave a channel.

At the Derry Farm dredge near Leadville, Colo., the method was to lift both spuds and to pull with the stern lines until the dredge was in the desired position, when the spuds were lowered. It was thought that a saving of nearly 10 minutes each time the dredge was moved resulted by adopting this method.

DUTIES AND REQUIREMENTS OF WINCHMEN.

In running a dredge care is taken to prevent the tailings from the sluices and stacker from so filling the dredge pond as to cause the boat to ground when it is being moved. At times it is difficult to retain sufficient flotation space, and an inexperienced winchman may ground the boat if gravel considerably deeper than usual is dredged, and if the tailing stacker is too short to dispose properly of the tailing, or if the fines are greatly in excess of the coarser material and the sluices are short the excess of tailing may ground the boat. A sand pump to dispose of fine material is occasionally necessary, and simple elevator buckets to deliver the fines from the side sluices to the conveyor belt were designed by Walter Johnson for a dredge in Alaska at which a large force of men were formerly required to shovel back the fines in order to prevent the boat from grounding on them. This plan resulted in a great saving of labor and increased the yardage.

The manipulation of the bucket line often accounts for a considerable difference in yardage handled and in the recovery of the gold content of the gravel.

The position of winchman is the most important on the dredge, and the yardage dug depends almost entirely on the winchman's knowledge of his work and the character of the gravel in different parts of the cut. It is important that the winchman be kept interested in his work for if he is lax in operating the bucket line considerable loss will result.

REGULATING SPEED OF BUCKET SWING.

On every electric dredge a wattmeter or similar recording meter, connected to the digging motor, should be placed so as to be easily seen by the winchman as a guide in his digging. It will tell him when his motor is pulling its regular load and will warn him of any unusual stress in the motor such as might be caused by running into a cave-in of gravel below the surface of the water which may not be visible above the water line. It will also warn him when his buckets are running out of gravel, and will help in many ways to maintain smooth operation.

In some ground it is possible to devise a constant speed of swing for the bucket winches, instead of letting the winchman wait for empty buckets to come up before swinging. Here a wattmeter is particularly useful, as it will show a decrease in load the instant the buckets are out of gravel. However, when bedrock is being cleaned, there is some question as to just how far a constant speed can be devised, but familiarity with the ground enables the careful operator to judge the proper time for moving the buckets to a new point of attack. To mention an instance, Walter Johnson was able to increase the yardage at the Bangor Creek dredge on the Seward Peninsula by 300 yards a day after a careful study of the conditions and after determining a proper speed of bucket swing. Similar efficiency has been obtained by the large California companies as a result of careful study, and the yardage has been considerably increased by having the buckets take smaller cuts and increasing the speed of swing. (See Pls. LVII and LVIII.)

In this connection Summers [a] makes the following statement:

> In the earlier days of dredging the rule was to drop deep and it was necessary to use the slow gear to pull the buckets through the gravel. It was found that this was wrong, as it put an unnecessary strain on bow lines, winches, and ladders, and in fact on every part of the machinery used in digging and elevating the gravel. The old cry in dredging was for more yardage, and as soon as a winchman had his digging ladder down to its limit he would swing to the other side of the cut and step up. In using low gear on the swinging it is possible to pull the buckets through the gravel faster than they can pick it all up, and in that way gravel was left in the bottom of the pond.
>
> As the buckets will hold a limited amount of gravel it is necessary to use only power enough to fill them, and this is most economically done by taking light drops and swinging on high gear. In this way it is impossible to pull through a lot of gravel on the bottom of the cut as the fast gear is not strong enough, and when it is stalled it is necessary to raise up enough to allow it to swing freely. In using the older method the power consumption was much higher than necessary and the results were poorer on account of more time lost owing to broken bow lines, buckets slipping off the lower tumbler, broken bucket lines, wear on bucket pins and bushings, and repairs incidental to the increased power strains.

On some dredges the superintendent has found it advisable to take the low-speed gear off the ladder winch, as the winchman when meeting an obstruction in hoisting would change to low speed and the resultant powerful pull would at times cause a break in some part of the machinery. Without the low-speed gear, he is obliged to relieve strain by reversing.

In this connection Summers [a] says:

> In stepping ahead on the smaller boats a 5½-foot step is usually about the best length in ordinary gravel. In using a much smaller step the boat is liable to surge back and forth. This is caused by the bucket engaging the gravel and passing beyond the point of its strain before the next bucket following engages the gravel. As soon as the bucket engages the gravel, it forces the boat back hard against the digging spud; then when the bucket has cut into the gravel beyond the point of its greatest strain, the boat will surge ahead until the following bucket will hit the gravel, and then the boat is forced back against the spud again. This condition continues unless a step is taken that is about twice the length of the pitch of the buckets; this results in two buckets engaging the gravel at all times. Speed of the bucket line is varied somewhat by conditions but ordinarily 60 feet per minute is the usual rate of travel. Side-swing speed and depth of drop also vary. Side swing is usually about 30 feet per minute and the depth of drop about 8 or 9 inches in solid gravel. Recent practice is to drop light and swing on high gear.

Regarding the causes of delays to dredging Jennings [b] presents the following table covering the work of dredges of the Conrey Placer Mining Co. in the Ruby, Mont., district:

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[a] Summers, M. L., personal communication.

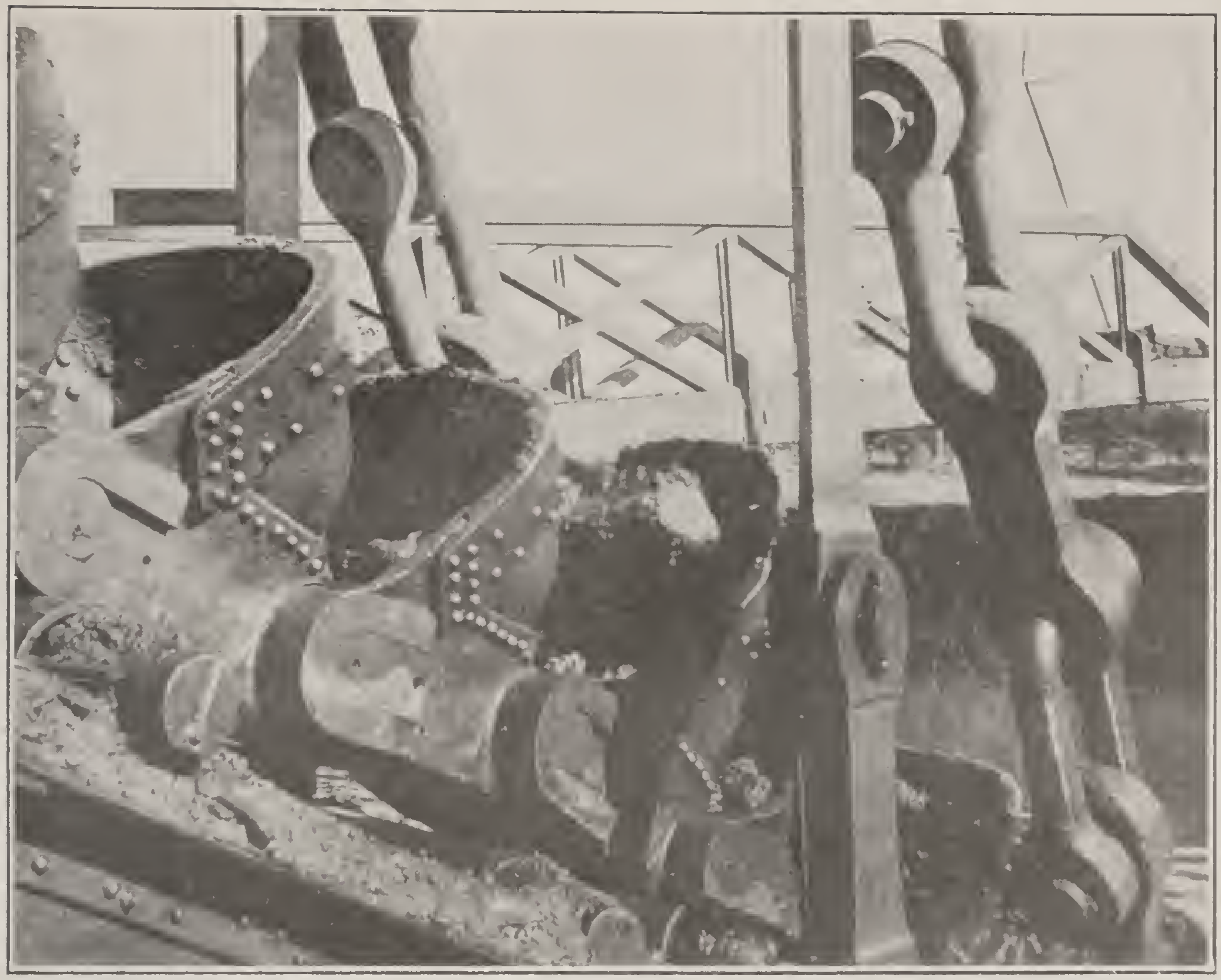
[b] Jennings, Hennen, History and development of gold dredging in Montana: Bull. 121, Bureau of Mines, 1916, p. 28.

Statement of running time and delays, Conrey Placer Mining Co., Dredges Nos. 1, 2, 3, and 4, Jan. 1 to July 31, 1915.

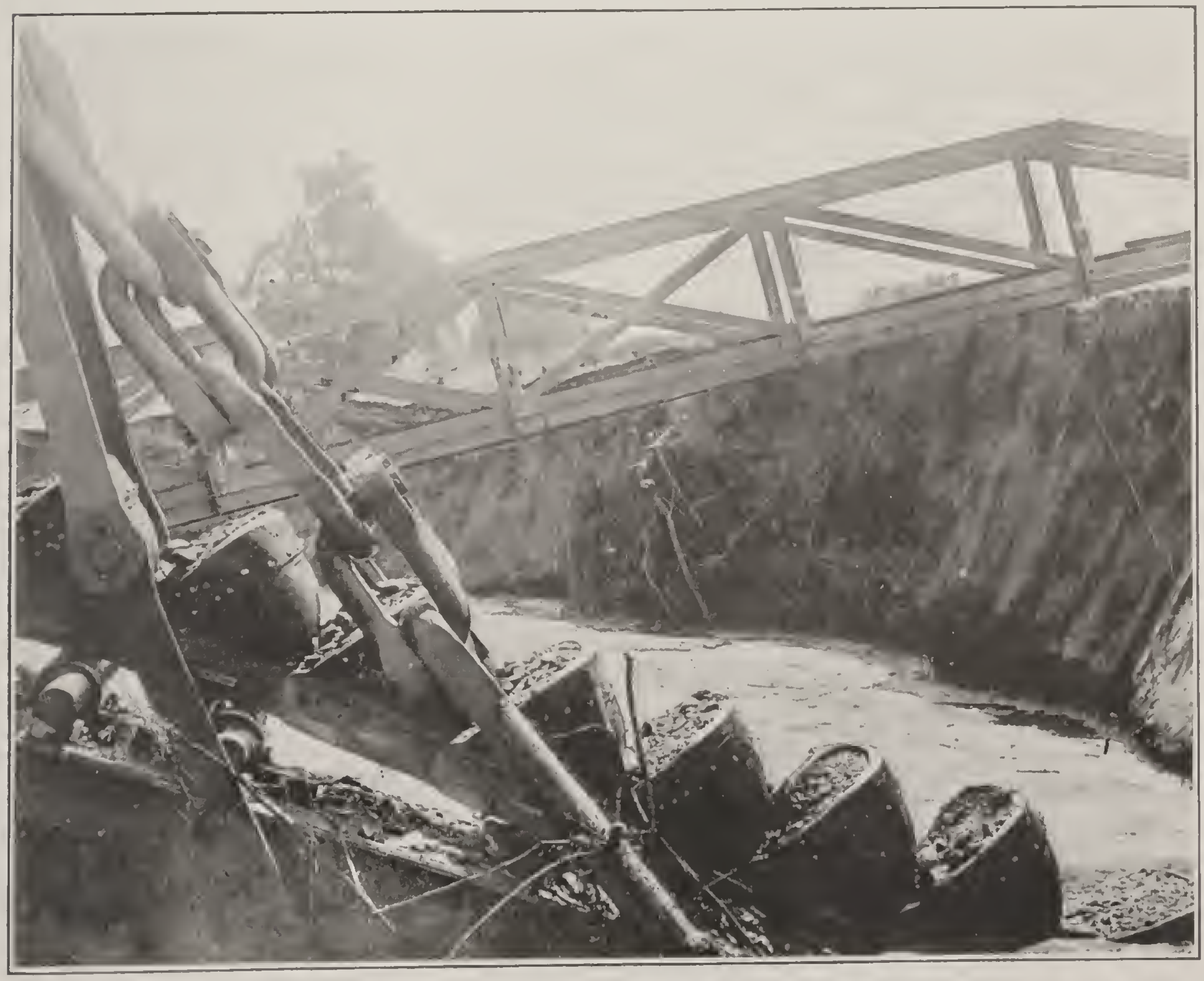
Cause of delay.	Time, per cent.			
	Dredge No. 1.	Dredge No. 2.	Dredge No. 3.	Dredge No. 4.
Clean-up	1.56	1.36	0.87	1.60
Moving ahead	3.89	6.33	2.75	3.80
Lines and spuds	3.42	3.01	2.25	2.95
Winches	.40	.41	.61	.13
Flume	....	....	.48	....
Ladder	.02	.14	.23	1.07
Tumblers	1.02	1.58	.70	5.91
Hopper	.16	1.24	....	.87
Grizzly	3.99	3.25	3.03	3.41
Bucket chain and drive	.62	3.05	2.52	2.27
Pumps and piping	.67	2.05	.38	.49
Electrical appliances	1.36	3.13	1.13	1.09
Heating system	....	....	....	.03
Débris	3.66	2.68	.36	.11
Power off	.68	.66	.53	.57
Stacker	.85	1.48	....	.41
Tables and sluices	.01	.39	.07	.97
Well	.02	.08	.18	.85
Gang plank	....	.06	....	.02
Unloading coal	.09	.15	.02	....
July 4	.71	.78	.78	.71
Hull and housing	.07	1.00	....	.01
Low water	.03	.38	....	....
Winter conditions	....	....	42.46	....
Funeral, President J. J. Myers	.01	.01	01	.01
Miscellaneous	.02	.01	.02	....
Total delays	23.26	33.23	59.38	27.28
Total running time	76.74	66.77	40.62	72.72
Total time	100.00	100.00	100.00	100.00

PRACTICE AT RUBY, MONT.

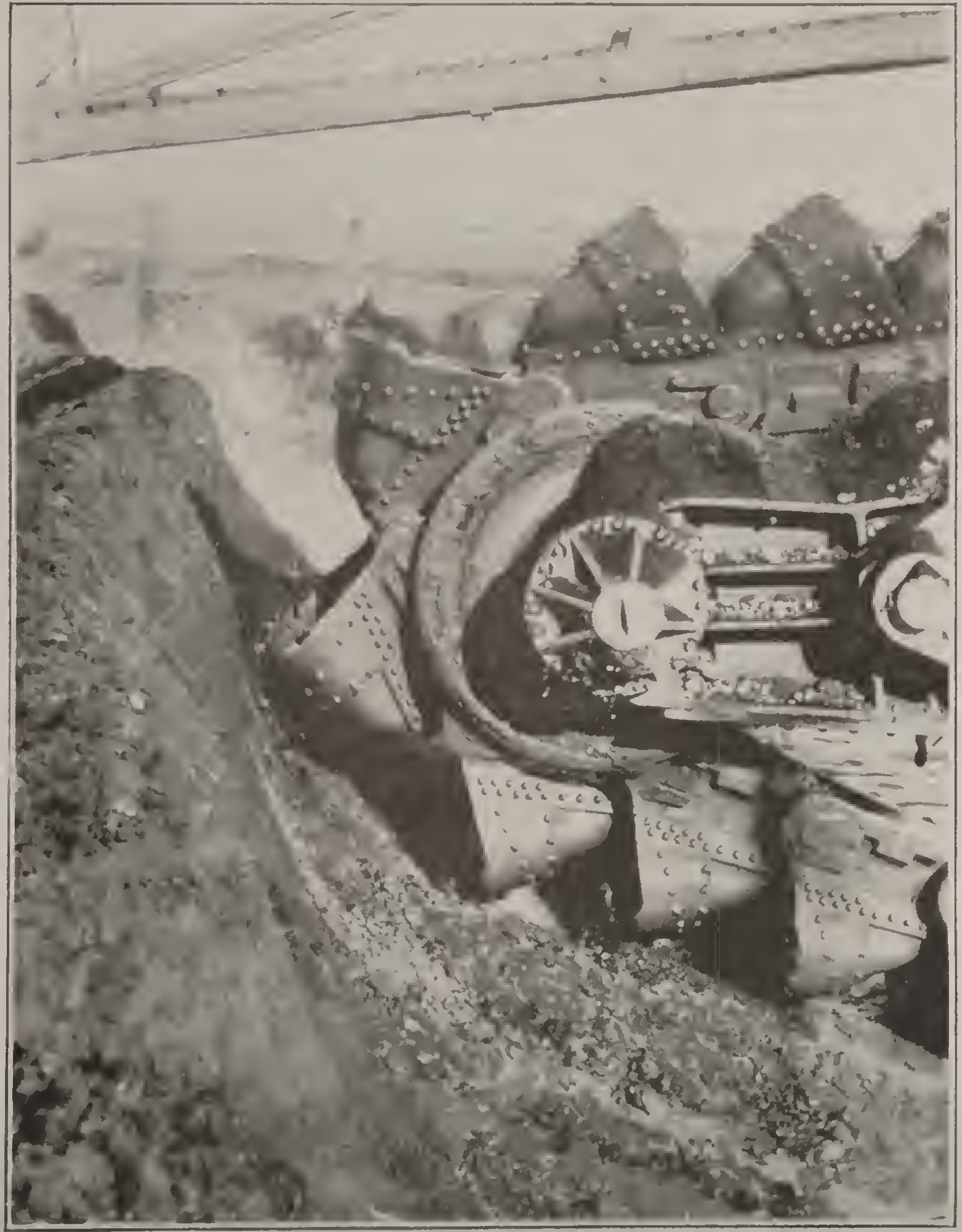
At Ruby, Mont., where the big No. 4 dredge is working there is about 16 feet of overburden and loose gravel. In digging this the bucket chain is run at the limit of 22 buckets per minute and the buckets can be readily filled at that speed. Below 16 feet the gravel begins to get very tight and is mixed with considerable clay, and the buckets are not filled when running at a speed of 22 buckets a minute; hence the speed is reduced to 20 buckets a minute; and near bedrock where the gravel is still tighter the buckets are run as slowly as 18 a minute, making an average of about 20 buckets a minute. It is found that it is better to reduce the speed of the buckets in hard digging and to fill them than it is to keep up the speed and to fill the buckets only half full, as the yardage handled is just as great, if not greater, with considerably less wear on the buckets and tumblers. It is necessary to dig up 6 to 12 inches of bedrock, depending largely on the character of the bedrock. If the bedrock is uneven it is necessary to dig deeper in the higher spots in order to be sure to get all the gravel in the depressions. In the Yukon along the Klondike River the gold in places extends into the bedrock for 2 or 3 feet whereas the upper gravels contain little or no gold. The bedrock is a soft decomposed mica schist and is easy to dig.



A. PARTLY FILLED BUCKETS, HARD GROUND.



B. FULL BUCKETS, LOOSE GROUND.



A. DIGGING HARD CEMENTED GRAVEL.

Note shape of face of cut and buckling of bucket line under tumbler.

B. LARGE BOWLDERS DUG AT NATOMA.

BUCKET-CLEANING METHODS.

The Natoma No. 8 dredge has worked in ground composed largely of sticky mud which can not be easily dug or readily sumped from the buckets. In addition the water in the pond becomes so muddy that it blocks the pumps, adds to the cost of operation and repairs, and reduces yardage materially. To overcome the sticking of clay to the buckets, an additional high-pressure pump has been placed on the dredge to deliver water through two nozzles, which are directed into the bucket as it is turning over the hopper. This arrangement proves of considerable benefit, and although it requires 250 additional horsepower it shows a net saving per cubic yard. A comparison of yardage records and power costs in 1913, before the change, with those in 1914, after the change, is particularly interesting.

Yardage record and operating costs per cubic yard, Natoma No. 8 dredge.

Year.	Time dredging.	Depth dug.	Material dug.	Material dug per hour.	Costs per cubic yard, cents.							
					Labor	Material.	Power.	Water.	Total.	Repairs.	General taxes, insurance.	Total.
	Hours.	*Feet.*	*Cubic yds.*	*Cu. yds.*								
1913........	6,616	62	1,807,235	271	1.01	23	1.49	14	2.87	2.81	86	6.34
1914........	6,370	67	2,053,568	322	.84	15	1.32	10	2.41	4.01	72	7.14

In 1914, as compared with 1913, the dredge dug 50 yards more per dredging hour, and consumed less power, with a running expense of 56 cents less per cubic yard, though the total expenses increased by 0.80 cent over those of 1913 on account of the necessary improvements, changes, and repairs.

On the hopper of the Viloro dredge, Oroville, to cut the clay from the buckets it was necessary to install two 1½-inch nozzles, fed by a two-stage Byron-Jackson pump delivering 1,200 gallons a minute and driven by a 100-horsepower motor. To prevent splashing the hopper was built up on the sides and covered. A part of the ground worked by the Viloro dredge contained 5 feet of indurated clay above the pay gravel, and underlying 5 feet of top soil. To facilitate dredging, the top soil was first removed by the bucket. Auger holes 8 feet apart in lines 5 feet from each other were then put down in the clay and blasted. The clay which prior to blasting was difficult to handle could then be easily dug. The cost of the extra work included the time of three men at $2.50 a day and three to five boxes of powder at $4.50, or an average of about $25 a day extra, but the extra expense was more than justified by the increased yardage handled. When the Yuba dredge was handling a stratum of clay that contained no gold, at a depth of 45 feet to 60 feet below water, an iron plate was temporarily used to block the hopper and the material was dumped into the well hole and not sent to the screen and tables.

In digging stiff clay, the dredge master generally finds it good practice to take light cuts, as buckets dump cleaner than if deep cuts are taken, when the buckets clog and the clay may also get lodged in the hopper.

When the Isabel dredge was handling a larger quantity of barren clay in Calaveras County, Cal., an improvised clay cut-out worked satisfactorily. It is described by Winston.[a]

The principle of the cut-out is shown in figure 16. A deflecting plate, *a*, operated by a lever, *d*, causes all fines from the shaking screen with which the dredge is equipped to pass either to the gold tables or, when the operator so desires, to a separate clay sluice, which extends a considerable distance beyond the stern of the dredge.

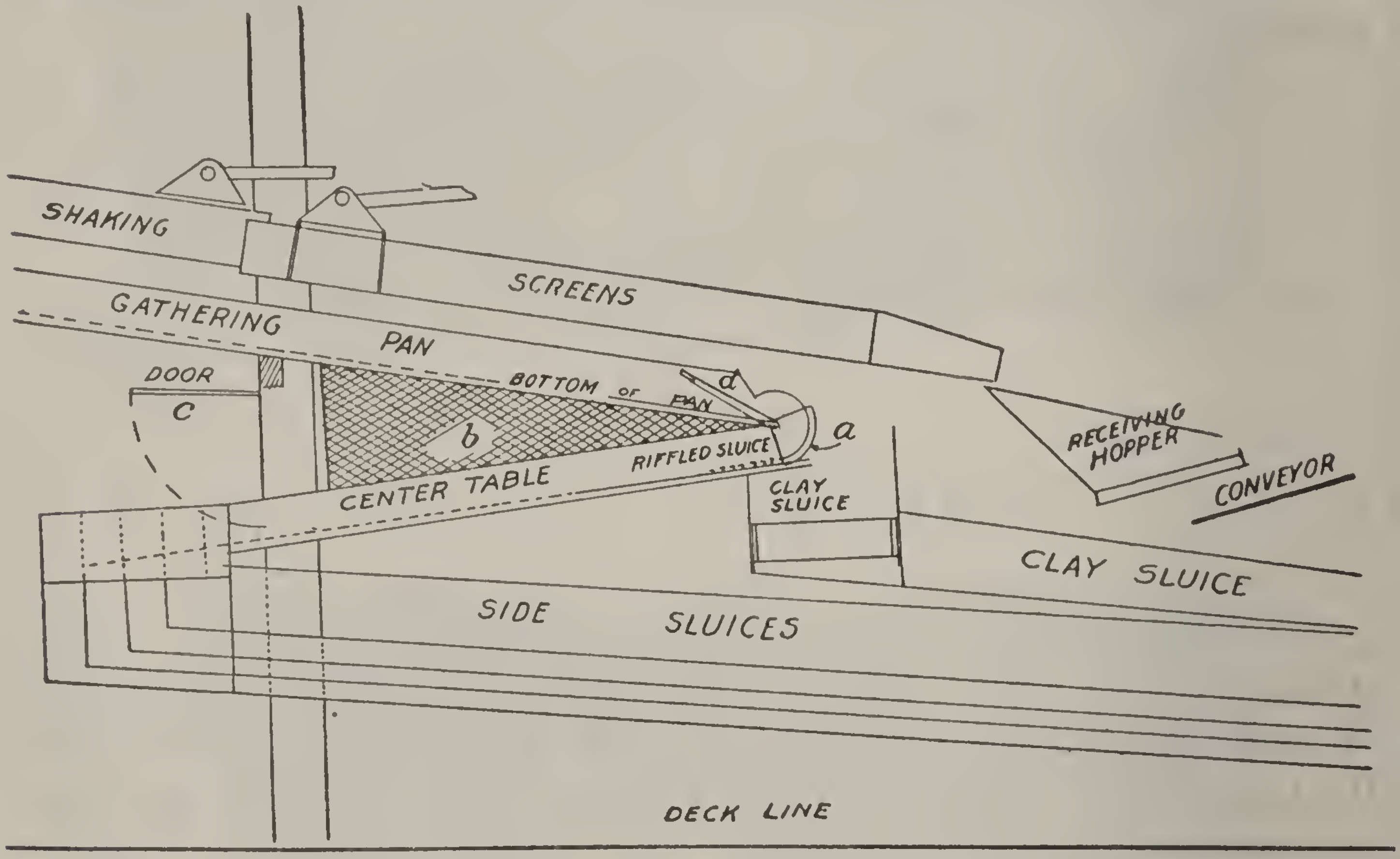


FIGURE 16.—Clay cut-out device for gold dredge.

Large lumps of clay of course are not broken at the screens, and pass directly to the tailing conveyor. The device shown, with properly placed hopper sprays for cleaning clay from the dumping buckets, has obviated most of the delay and reduced the gold losses due to clay robbing the sluices.

USE OF MONITORS AND OF BLASTING IN CEMENTED GRAVEL.

In handling hard, compact gravel, monitors were placed on the front of some dredges, especially in the Folsom district. Although these were of some use in washing down high banks of gravel of a medium compactness, they were of little use against cemented gravel. A number of years ago at Oroville blasting was tried with considerable success in handling cemented gravel, but the extra cost was 2 to 5 cents a cubic yard. The method of operation was to drill holes 5 to 15 feet into bedrock and placed 25 to 250 feet apart. Charges of 50 to 120 pounds of 40 per cent dynamite were used in the holes, the charges being fired either by electricity or by fuse. Blasting in front of the Natomas dredges, which were dredging in hard cemented material, did not give very satisfactory results, as the gravel broke into large chunks which went through the screen without disintegrating.

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[a] Winston, W. B., Clay cut-out, Isabel dredge: Min. and Sci. Press, vol. 101, Dec. 24, 1910, pp. 838–839.

LIFE OF HULL.

The life of a wooden hull in California is generally estimated at 10 years for small dredges and 7 for large ones, and a steel hull for either size is expected to last 10 years. Wooden dredge hulls have lasted longer than the period mentioned, as, for example the old Continental, which lasted more than 10 years, the Pennsylvania, which is still working, though built more than 13 years ago, and the Viloro hull, which was 11 years old when the dredge was burned in October, 1914. At Breckenridge, Colo., the hull on the No. 3 dredge is 15 years old and is seemingly still in good condition. On account of climatic conditions there has been no evidence of dry rot, and when it was necessary to overhaul the hull about the stern no evidence of dry rot was found. This condition is not usually found in dredge hulls. A dredge hull is ventilated by means of blowers at the stern of the boat, which force fresh air through the hull, the air coming out through the ventilators on the bow. Such ventilation is essential for dredges operating in a warm or humid climate, and neglect to ventilate the hull properly has caused serious damage on more than one dredge. The general practice is occasionally to paint the interior of the wooden hull with hot crude oil, which considerably preserves the life of the timbers.

On dredges operating in cold climates, a steam-heating plant is arranged either by having the boiler placed in the hull or on the main deck, and by having coils of steam pipe placed around the interior of the dredge and at suitable places in the housing and stacker.

Few dredges work throughout the winter in regions where that season is extremely cold. As exceptions may be mentioned Dredges Nos. 1, 2, and 4 of the Conrey Placer Mining Co., at Ruby, Mont., where heating by steam pipes is employed. The steam circulates under the gold tables and along the stacker, which is supplied with a canvas housing. Electric heaters are used on some dredges with success. Miles[a] describes the method of operating the Boston &

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[a] Miles, J. H., Winter dredging in Idaho: Min. and Sci. Press, vol. 108, Mar. 14, 1914, pp. 455–456.

Idaho dredge in Idaho during the coldest weather. It was afterwards found advisable not to operate during the winter months.

REPAIRING AND MAINTAINING HULLS.

Dredge hulls frequently require repairs, either near the stern where the spud strain develops, or near the bow when the weight and strain on the front gantry causes a springing of the hull and starts leaks. On one dredge the hull was repaired by pouring a rich mixture of cement between a bulkhead formed by the side planking of the dredge and timbers placed on the inside of the hull frame. Generally the hull is repaired by building in solid with new lumber against the inside shell of the boat, though at times it is necessary to pump all the water from the pond, and make extensive repairs on the hull, especially if the dredge has capsized.

Steel hulls also have given trouble at spud casings and made dry docking and extensive repairs necessary. Consequently, these parts have been strengthened and heavy spud-casing springs have been placed both on the hull and on a level with the upper deck, to take off much of the strain caused by pressure against the spuds during digging.

On nearly all dredges, arrangements are made to handle any leaks that may develop in the hull, by having either separate emergency pumps or pump suctions attached or attachable to the monitor pump. Some operators have a float so arranged that a bell will notify them if water rises on the hull, whereas others trust to regular investigations. On the smaller dredges a hand pump on the bow generally handles all ordinary seepage, although there is also an emergency pump for use if a leak develops.

On some of the older dredges continual pumping is required in order to handle the water coming in the hull, and a number of dredges have capsized owing to leaks caused by sudden strains to the hull planks. Although all operators appreciate the need of properly ventilating the hull, some believe that it is unnecessary to pay much attention to the dredge housing, as wood housing will ordinarily outlast a hull two to one.

Most operators take pride in the appearance of their boats, and paint the housing at least once every two years; others, especially on headline boats, or boats having small flotation and on those handling considerable clay, feel that it is useless to try to keep the dredges clean. Opinions regarding the care given machinery also differ, and one successful dredge operator feels that any unnecessary care given to the machinery merely to improve its appearance without adding to the operating efficiency of the dredge, is useless. The larger companies make it a rule to keep the dredge and machinery as clean and neat as possible. The men take a much keener interest in the work if everything is kept clean and tidy. Some companies have notices in conspicuous places, "A place for everything and everything in its place." After a breakdown, there is always more time lost if there is a mess of grease and grime that has to be removed before the work can be properly done. Most men have an aversion to greasy muck, and dread starting a repair job in the midst of a lot of unnecessary dirt.

HANDLING OF LARGE BOWLDERS.

The handling of large bowlders in dredging is always somewhat difficult. The method used on the dredge of the Andrada mines (Ltd.), in Portuguese East Africa is worthy of mention. When a bowlder too large to be lifted is encountered, it is usually either passed over or is pushed to one side; but in ground containing many medium-sized bowlders, the bowlders must be raised in order to recover the mineral content of the ground, especially when the bowlders are near bedrock. In 4⅔ months, from February 10, 1914, to the end of June of the same year, the Andrada dredge, a close connected 7½-cubic foot boat, treated 547,700 cubic yards in a ground full of bowlders and of clay, and picked up 4,400 bowlders weighing on an average of more than a ton each.

Elimination of bowlders from the buckets is the most delicate part of the work. The ordinary method in dredging is to stop the bucket chain, and, after tying the bowler with slings, to raise it with chain blocks suspended to a moving crane, and to set it on the deck. When the bowlder is very big and is squeezed in between two buckets, this operation requires at the least 10 minutes. By this method the Andrada dredge has pulled up more than 200 bowlders of an average weight of one ton, and the time lost has averaged just about 10 minutes a bowlder.

In February, however, the number of bowlders found became so large, that the removing of them by crane, operated at first by hand and later by winch, caused such loss of time that the actual dredging fell to about 50 per cent of the capacity of the boat. At that time cracking the bowlders in the buckets by blasting, and discharging the pieces through the screen and stacker was tried. A small dynamite cartridge was placed on the bowlder, covered with clay, and fired. The bowlder was not broken in small pieces, which flew and injured the dredge, but was only cracked and caused to settle in the buckets.

During March, 1914, the Andrada dredge picked up 1,510 bowlders, fired 1,544 dynamite cartridges, and lost only about 1,800 minutes, or, say, 60 minutes a day, or 1.2 minutes a bowlder. During the month the dredge treated 131,000 cubic yards of ground averaging 15 feet deep. In four months the chain of 62 buckets withstood 4,600 blastings (nearly 72 blasts per bucket), and only 5 buckets were pierced in the hood. The holes then made were only about 5 inches diameter and were easily patched and did not interfere with the working of the buckets.

The blasting of bowlders in buckets can be undertaken without danger, provided care is taken as to the quantity and placing of the dynamite used and the buckets are strong enough.

NIGHT LIGHTS.

In working at night, either arc lamps or clusters of incandescent lamps are used, the arc light being the more economical. In addition, it is good practice to provide the dredge with a searchlight placed on the front gantry and operated from the pilot house. This searchlight facilitates the work of the winchman and furnishes light for shore work, such as moving power cables and shifting side lines. After accidents to the electrical equipment the large acetylene lights have been found useful in dredging operations.

METHOD OF MAKING REPAIRS.

In California the small dredging companies run the dredges continually until these require a complete overhauling, when they are shut down for 10 days or more while the necessary repairs are made (Pls. LIX, *A*, and LX, *A*). The large companies, and some of the small operators have a separate repair gang which is always engaged on the boat during clean-up periods, when minor repairs, such as changing bucket lips and tumbler bearing plates are made. Extensive overhauling is often started during clean-up periods, on the principle that repairs should be made as soon as needed. The repair crew of the large California companies consists of eight or nine men, who soon gain a knowledge of all the different methods of repairing used on the different dredges. The maintenance of a crew of trained repair men greatly shortens the time lost on shutdowns. The dredge masters are encouraged to try new methods for hastening any detail of repairing or operation, and in that way there is an incentive for every man to try to improve on the methods in use.

In the northern fields, as pointed out by Perry,[a] all the heavy work is done during the winter or closed season. In the autumn, when work is suspended, the dredges are stripped of their bucket lines and conveyors. The buckets, screens, pumps, and all parts subject to heavy wear are taken to the shops and thoroughly overhauled. The dredges are then put in as good order as possible, with a view of running them continuously throughout the season of approximately four months. During the operating season all repairs upon the dredges are minimized. For bucket-line repairs the whole line is

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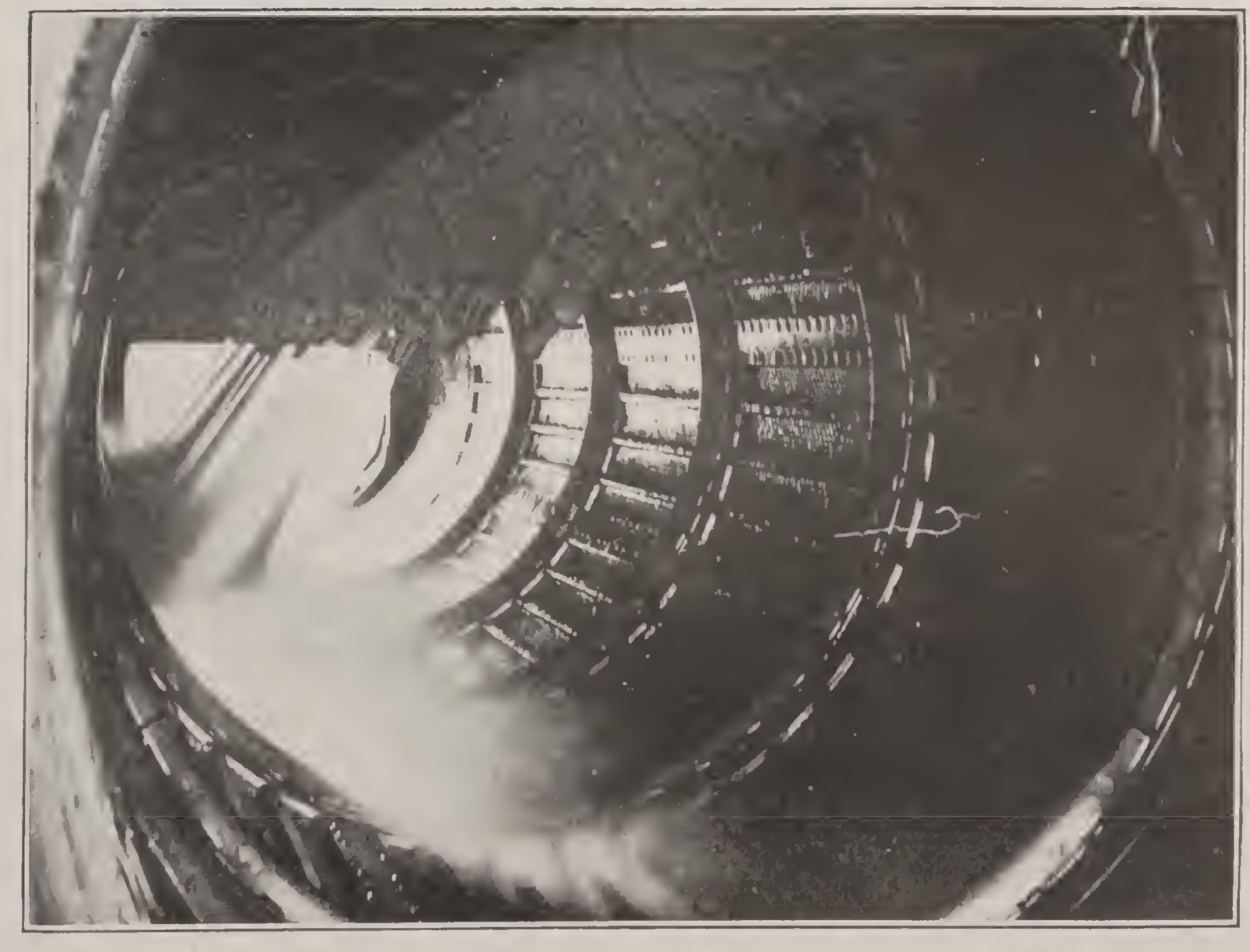
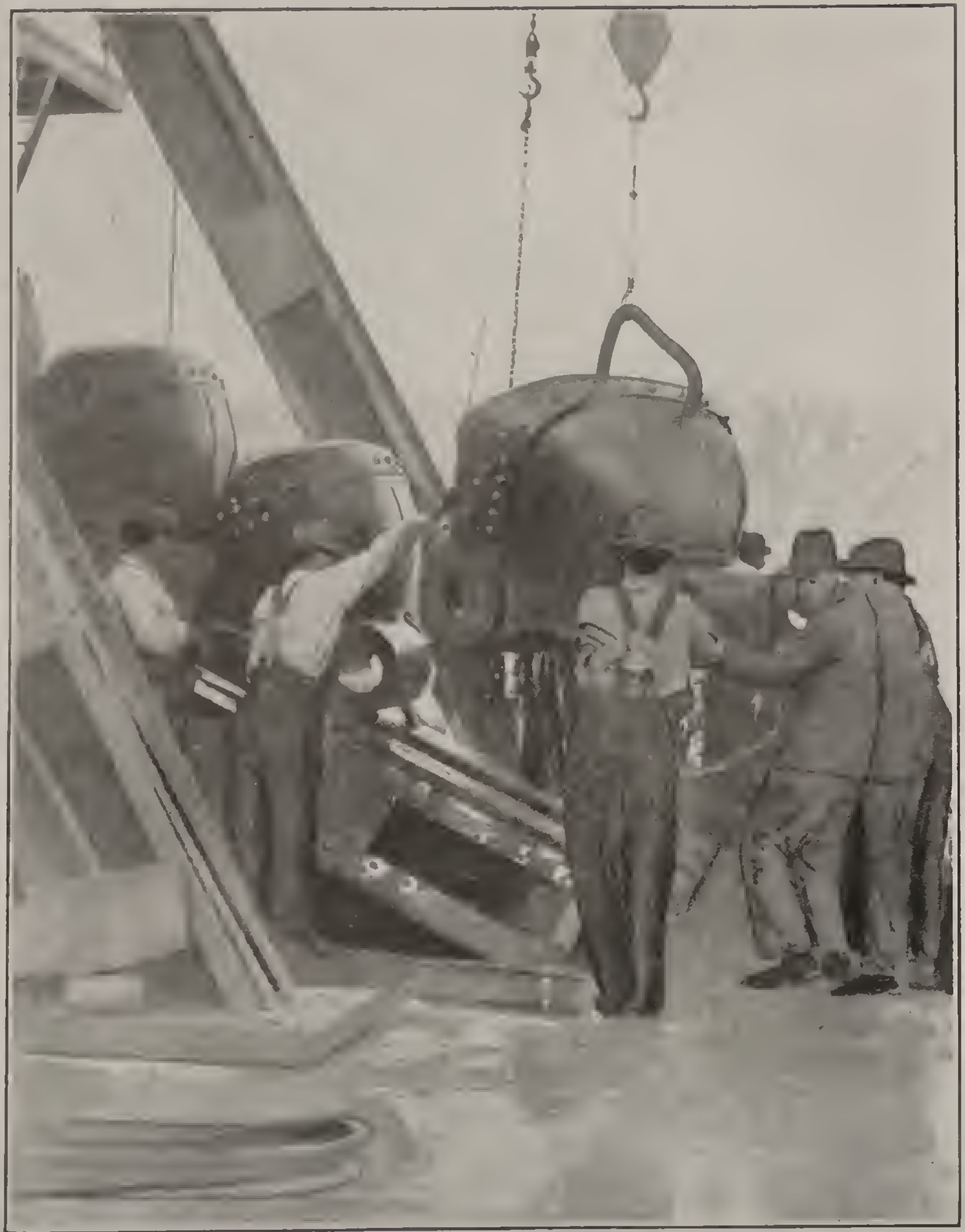
[a] Perry, O. B., Development of dredging in the Yukon Territory: Trans. Canadian Min. Inst., vol. 18, 1915, pp. 32–34.

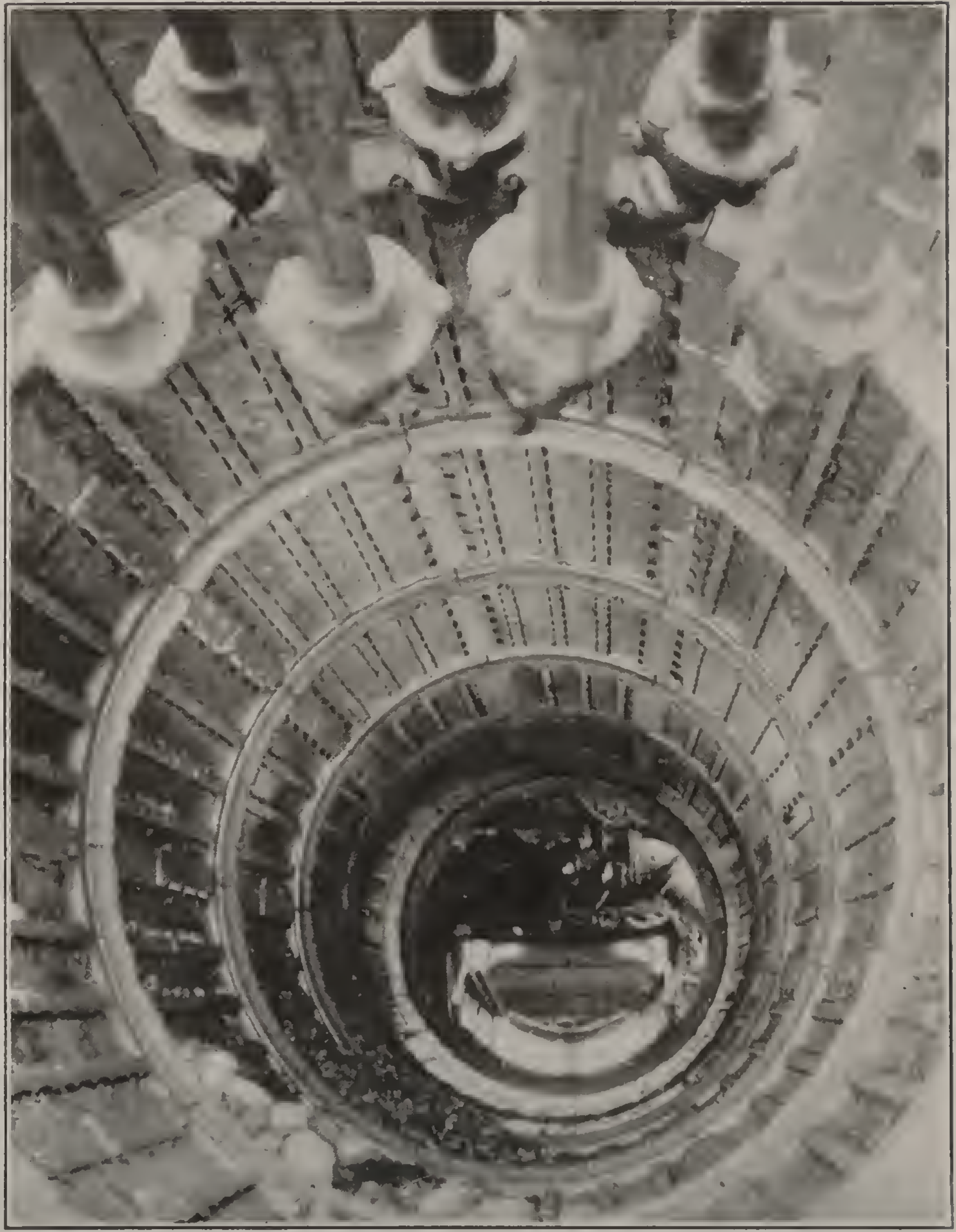
A. TROUBLE WITH THE BUCKET LINE.



B. WATER NOZZLES IN PIPE IN INTERIOR OF REVOLVING SCREEN.



A. CHANGING BUCKETS.



B. INTERIOR OF A REVOLVING SCREEN, SHOWING END PLACING OF WATER NOZZLES.

sometimes removed and is replaced by a line that has been entirely overhauled, the other line being repaired for the next replacement. This arrangement illustrates the advantage to an operating company of having all boats of the same size with interchangeable parts. The repair crews of some of the large companies when renewing bucket lips use a portable air compressor which they take to the different dredges. This method is much quicker and cheaper than taking the buckets to the repair shop.

Formerly at the Natomas Consolidated there were several different sizes of dredges, each dredge working under different conditions; and nearly every dredge even of the same capacity was built differently and by different companies. Consequently, it was necessary to carry a large stock of supplies. Under new management this disadvantage is being adjusted, and any dredges rebuilt are made in two capacities only, machinery parts being made interchangeable for all dredges of the same capacity. This plan has resulted in an appreciable reduction in working cost.

Broken or badly worn buckets are ordinarily changed as soon as possible or at the first clean-up period. If new parts are not available, or if for other reasons delay seems inadvisable, the line may continue running even with a number of broken buckets. The method of changing buckets varies in different fields. On some dredges the old line is first removed entirely, and the new one added in sections which are first connected on the bank, and are hauled to the ladder by cables from the main winch. Sometimes the buckets are replaced one by one, new buckets being transferred to and from shore by cable. The practice at the Yuba Consolidated is to change buckets in sets of six or seven; that is, the extra buckets are taken to the dredge and replace the same number of broken or worn ones in the line.

On the No. 14 dredge the dredge crew directed by Mr. Williams, the dredgemaster, took off and replaced six 16-foot buckets in 40 minutes. It required 12 minutes only for putting on the new buckets. One hundred and eight 7½-foot buckets were taken out and replaced in 11 hours and 15 minutes. The Yuba company handles buckets by a special small hoist operated by a water wheel fed by a pump, and a small hydraulic hoist is attached to the front gantry to handle bucket pins. Swinging rams, hung on either side of the ladder on the front of the dredge, are used to knock out old bucket pins and drive in new ones.

USE OF DEPTH INDICATOR CHART.

On some dredges a time chart is so arranged by connections to the digging ladder that when the ladder is raised or lowered the depth of digging is indicated (fig. 17).

On the Conrey dredges an indicator is arranged on the upper tumbler shaft automatically to keep count of the number of buckets dumping.

Figure 18, reproduced from a sketch furnished by Charles Kammerer, illustrates the method of ascertaining the yardage dug. It also shows the method developed at Ruby of returning the bowstring lines to a sheave on the lower end of the ladder, before passing to the swing winch, thus taking much of the strain from the bow of the hull. (See chapter on "Operating Cost.")

On many dredges working in a known channel, and making only one cut, panning is practiced to determine the gold content of the ground. Sometimes a small section of the gold table is cleaned daily, and the course of the dredge is changed as results indicate advisable. At Breckenridge a short distance on either side of the richer section being dredged, the gold content falls below operation costs, but in order to insure working to the extreme limit of the good ground, the cut is extended slightly beyond the known limit of the channel.

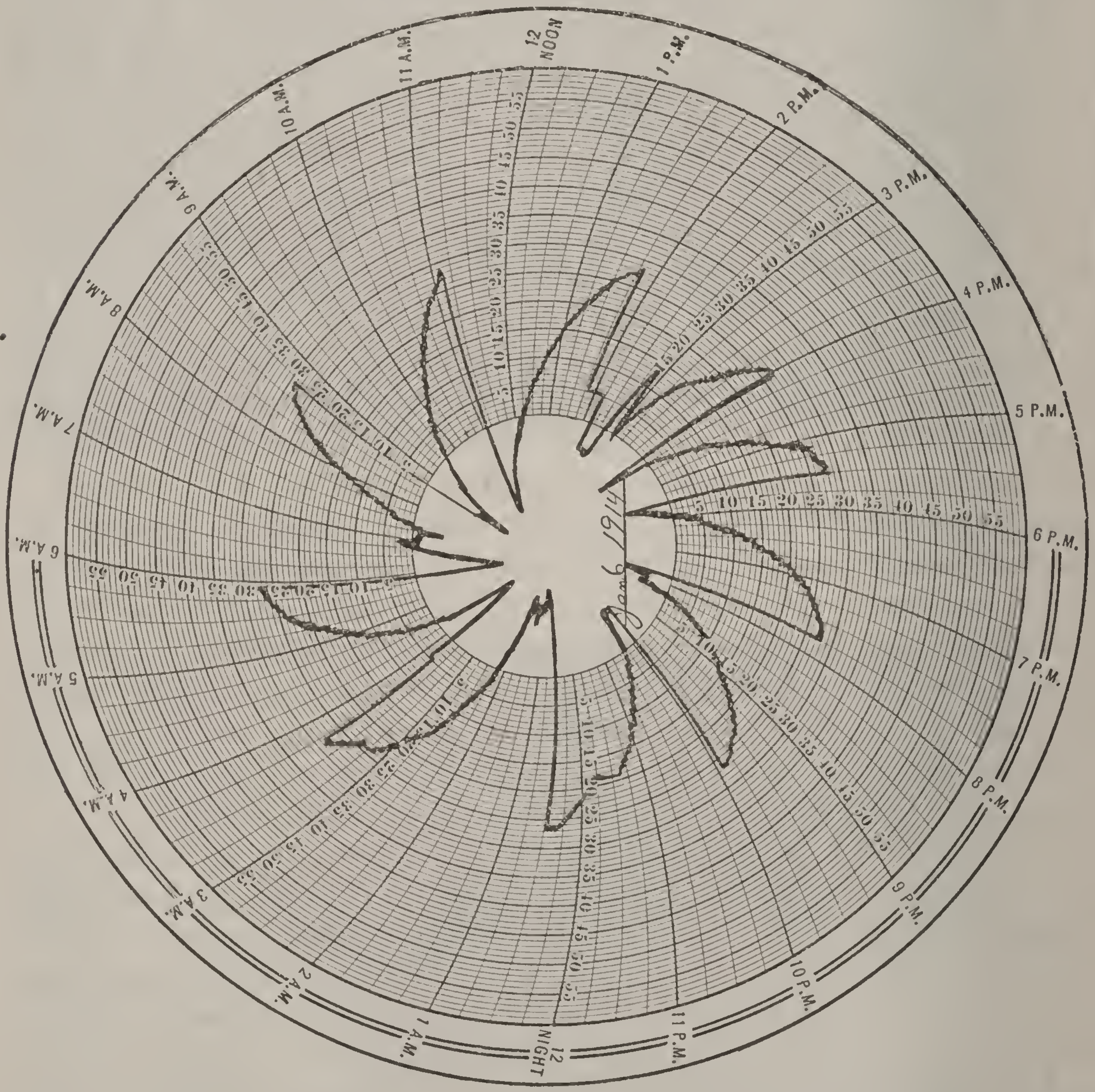


FIGURE 17.—Specimen chart showing record of depth dug by gold dredge.

EXTENT AND CAUSES OF LOST TIME.

When the dredge is in operation, all time not actually spent in digging is generally considered as time lost. Some operators do not consider stepping ahead as lost time, although time spent in clean-ups is nearly always so regarded. The greatest proportion of lost time is due to accidents, delays, and repairs to the digging equipment, including the changing of buckets and the renewing of bucket tips, tumbler wearing plates, etc., but changing of the cushion type upper tumbler plates formerly used took five men 10 hours. The present type of upper tumbler obviates most of this loss of time.

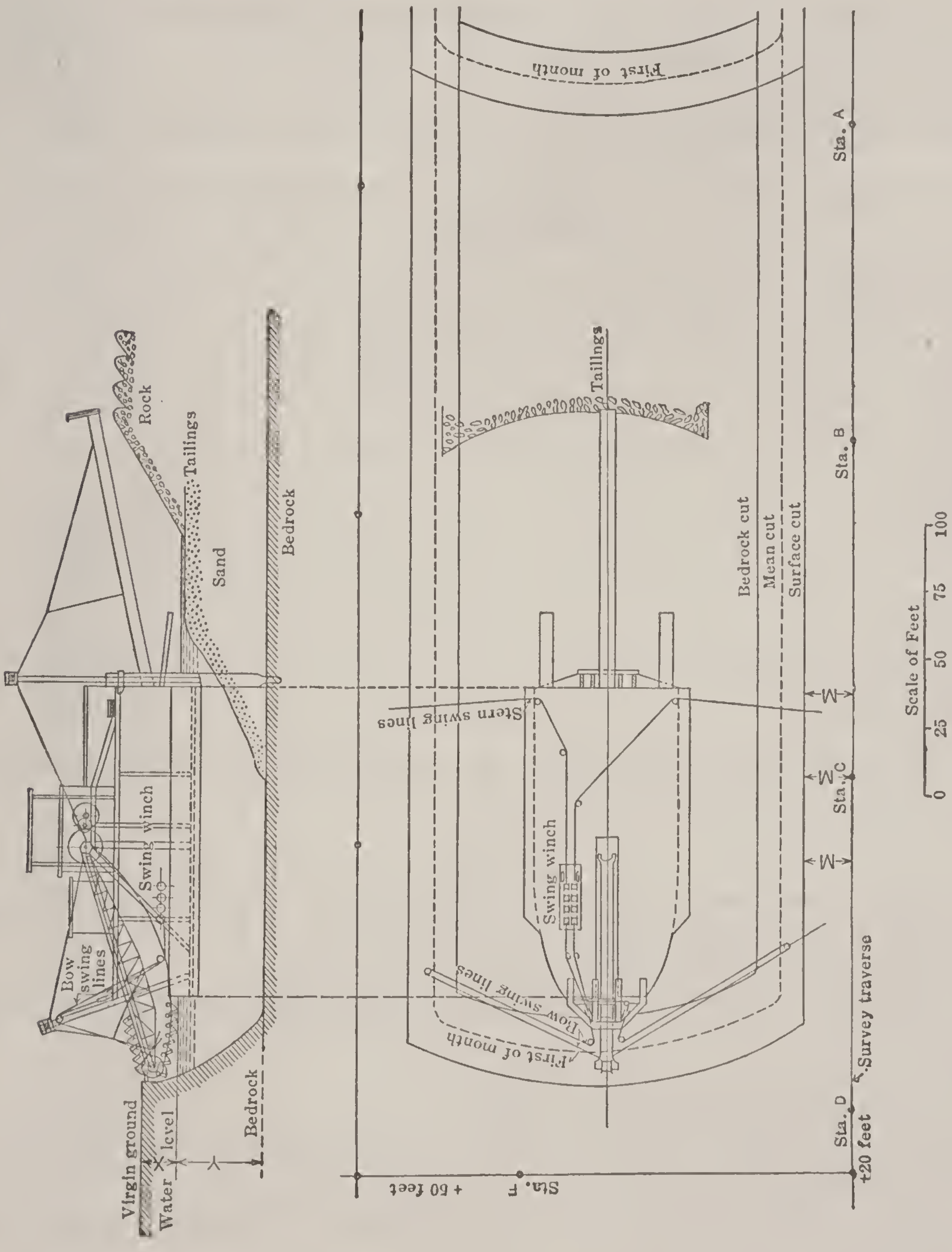


FIGURE 18.—Plan and elevation of 9-foot dredge, showing swing winch and swing lines; also method of calculating yardage handled.

The shutting off of power because of storms and other causes outside the dredge master's control sometimes causes considerable loss of time, though by most operators this is not considered loss of operating time. They merely deduct this time from the total time the dredge was in operation. Other parts of the dredge that require frequent attention and are responsible for loss of time are the screen and the tailing stacker, and also the pumps, particularly when the pond water is muddy.

The table following shows the percentage of lost time attributed to various parts of the machinery in 1914 on dredges 3 and 4 operated by the Marysville Gold Dredging Co. near Marysville, Cal. Loss of time on other dredges is shown in different tables given in the section on working costs. The total lost time on all Natomas Consolidated dredges for 1914 averaged 18 per cent; on all dredges of the Yuba Consolidated Gold Fields, 14 per cent; and on all dredges of the Marysville Gold Dredging Co., 15 per cent of the possible time.

Delays in dredging referred to causes, dredges 3 and 4, Marysville Gold Dredging Co., California, 1914.

Cause of delay.	Per cent of total delay.
Stepping up	13. 6
Stacker	4. 2
Clean up	11. 7
Motors	1. 0
Lower tumbler	8. 2
Upper tumbler	4. 4
Lines	5. 1
Pumps	3. 1
Screen	7. 8
Buckets	8. 1
Power	6. 6
Rollers	4. 6
Sand	. 2
Winches	2. 0
Upper tumbler drive	2. 9
Miscellaneous	16. 5

The following notice regarding the recording of delays is issued by the Marysville Gold Dredging Co.:

NOTICE.

In order that reports may be uniform, please be guided by the following when making record of delays on winchman's shift report:

Stepping up.—Include also moving dredge to new position.

Stacker.—Includes stacker belt, stacker drive (not motor trouble), idlers, guy ropes, drums, etc.

Clean-up.—Time actually engaged in clean-up.

Motors.—This refers to all motors on dredge.

Lower tumbler.—Refers to time lost oiling and any repairs to lower tumbler, or lower tumbler bearings or bushings.

Upper tumbler.—Includes all parts of upper tumbler and bearings of same, and upper tumbler shaft.

Lines.—Refers to ladder lines, swinging lines, or stern lines.

Pumps.—All delays caused by pumps (not pump motors), including also screen around intake.

Screen.—Delays on account renewing or repairing revolving screen, screen drive, rollers, bearings, driving gears, spray pipe, etc.

Buckets.—Time lost repairing or replacing; also when off tumblers, unless clearly caused by defective tumbler.

Power.—Delays caused by power being off or low voltage.

Rollers.—Refers to ladder rollers, bearings, chairs, bushings, oiling, etc.

Winches.—Refers to both ladder hoist winch and main winch for swinging lines.

Bucket drive.—Refers to all gears, shafts, bearings, etc., in connection with driving upper tumbler, including magnetic clutch.

Electric equipment and power lines.—Refers to power cables, power lines, switches and switch boards, controllers, transformers, wiring, and all electric equipment except motors.

Spud.—All delays caused by spuds, springs, spud housing, lines, etc.

Hopper.—Delays caused by clogging of hopper or any repairs to hopper.

Ladder.—Digging ladder only; cleaning, repairing; includes ladder arms and chafing beams.

Tables.—Includes also sluices, distributors, and save-all.

Miscellaneous.—All other causes of delays not covered by the above headings, giving cause.

In making bucket-line report, when giving number on base from which a pin has been removed, or put in, use the number on base in which the lug fits.

The following table [a] showing segregated causes of lost time on a small dredge in Alaska is of interest, as it is not only typical of the work of a single dredge in an isolated field, but of a modern, steam-driven dredge operating under severe conditions in the northern field. Out of a total possible dredging time of 3,144 hours, 805 hours, or 25.7 per cent, was lost for various causes.

Causes of lost time on an isolated gold dredge in Alaska.

Cause.	Time lost.	
	Hrs.	Min.
Bucket line [b]	19	39
Lower tumbler		45
Upper tumbler	1	10
Main gearing	103	46
Digging ladder	24	25
Hoppers	33	40
Save-all	20	15
Screen	61	19
Stacker	75	54
Flumes, etc	27	31
Pumps, screen	70	37
Winches	60	53
Lines	64	5
Spuds	16	15
Digging engine	4	20
Pump engine	18	55
Oiling	6	31
Stepping	141	5
Clean-ups	35	10
Rocks	13	14
Fuel	2	0
Light plant		13
Frozen ground	99	45
Miscellaneous	144	24
Total	805	1
Thawing (included in "Miscellaneous")	79	5
Bringing water to pond (included in "Miscellaneous")	29	0

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[a] From report of Gerald Hutton, dredgemaster for C. J. Berry Dredging Co., 1915.

[b] Most of the lost time was caused by the bucket line coming off the lower tumbler during heavy side feeding.

The table following shows causes of delay and percentage of time lost in operation of dredges in Montana:

Causes of delay on dredges Nos. 1, 2, 3, and 4, of the Conrey Placer Mining Co. and Poor Farm Placer Mining Co., 1914.

Cause of delay.	Time lost on dredge No.—								Percentage of time lost on dredge No.—			
	1.		2.		3.		4.		1.	2.	3.	4.
	Hrs.	*M.*	*Hrs.*	*M.*	*Hrs.*	*M.*	*Hrs.*	*M.*				
Clean-up	131	50	119	15	97	30	153	20	1.50	1.36	1.11	1.75
Moving ahead	367	5	448	55	250		299	55	4.19	5.13	2.85	3.42
Lines and spuds	260	10	176		278	20	212	15	2.97	2.01	3.18	2.42
Winches	98	50	69	55	180	40	17	5	1.13	.80	2.06	.19
Flume					147	35					1.68	
Ladder	11	45	71	05	34	25	453	15	.14	.81	.39	5.18
Tumblers	86	35	184	55	166	55	313	55	.99	2.11	1.91	3.58
Hopper	4	10	33	40	43	30	73	50	.05	.38	.50	.84
Grizzly	212	25	161	25	279	55	115	20	2.43	1.84	3.20	1.32
Bucket chain and drive	95		277	55	92	45	275	35	1.09	3.17	1.07	3.15
Pumps and piping	61	35	47	15	55	5	27	25	.70	.54	.63	.31
Electrical appliances	43	55	72	35	284	20	337	10	.50	.83	3.25	3.85
Heating system		10		15		40		15				
Débris	212	55	108	35	219	30		55	2.43	1.24	2.51	.01
Power off	23	05	39	45	21	40	19	10	.27	.46	.25	.22
Stacker	186	05	90	55			63	55	2.13	1.04		.73
Tables and sluices	1	00	7	55		20	69	35	.01	.09		.79
Well	13	30	3	15	8	10	50	55	.15	.04	.09	.58
Gang plank	1	30						20	.01			
Unloading coal	8	10	16	25	17				.09	.19	.19	
July 4	32		40		40		40		.36	.46	.46	.46
Christmas	32		40		40		40		.36	.46	.46	.46
Scow	1	30							.01			
Hull and housing	4	30	4	50					.05	.05		
Low water					3						.03	
Winter conditions					1,024						11.69	
Raising No. 1 dredge	192	50							2.20			
Headframe			68	30						.78		
Miscellaneous				40	6	30					.07	
Total delays	2,082	35	2,084		3,291	50	2,564	10	23.76	23.79	37.58	29.26
Total running time	6,677	25	6,676		5,468	10	6,195	50	76.24	76.21	62.42	70.74
Total time	8,760		8,760		8,760		8,760		100	100	100	100

WATER CONSUMPTION IN GOLD DREDGING.

The quantity of water required in dredging varies with the size and type of dredge and the character of the material to be handled. The fresh water that runs into the dredging pond does not by any means indicate the quantity supplied to the gold-saving tables by the pumps, as the water from the dredge pond is constantly reused. The quantity of fresh water obtained from outside sources depends on the nature of the material to be washed and on the water level on the ground being dredged. The quantity of water that is sent to the gold-saving tables is regulated by the dredge pumps. The following calculations assume that the pumps were running at full capacity, and do not take into consideration excess water lifted by the dredge buckets. On a large dredge working in easily washed gravel there are two 14-inch pumps which, when the dredge is at work, furnish water as follows: High-pressure pump, 5,500 gallons per minute; low-pressure pump, 5, 500 gallons per minute, or a total of 660,000 gallons an hour. On the basis of a running time of 22 hours and a daily capacity of 10,000 yards, these figures would indicate approximately 10 cubic yards of material washed per inch of water, and about 1,400 gallons to a cubic yard of material dug and washed in the screen.

The percentage of fine material in undersize that goes to the tables from the screen varies from 35 to 60 per cent of the total material handled, so at this stage of the operations there is more water used per cubic yard of material washed than is shown by figures based on the total yardage dug.

As a rule dredges working river bottoms or owning a gravity water supply take no account of the fresh water used, but on dredges for which water must be purchased the quantity used is recorded by a gage. Figures taken from the report of the Natomas Consolidated for 1913 indicate that there was supplied an average of 100 inches of water per dredge for the Folsom dredges.

Dredges working in Calaveras County pump water from the river, the only cost being pumping charges. The quantity of fresh water supplied for the 5-foot boats is estimated to be 50 to 70 inches. At Bohanan Bar near Salmon, Idaho, the manager calculated that when the dredge was working upstream on a 4 per cent grade, 350 inches of water was required from outside sources to supply fresh water and to maintain flotation of the dredge.

On dredges like the Natoma Nos. 9 and 11, which work much sticky clay, in addition to the pumps supplying water to the screens and the tables, the quantity of wash water is increased. There are two nozzles discharging water at high pressure (910 pounds) into the buckets as they dump into the hopper. These pumps would require an increased water supply of about 50 per cent over that for the other dredges. The yardage of these dredges averages about 350 yards per hour; hence the water furnished the screen would be about 2,800 gallons per cubic yard of material dug.

Plate LIX, *B* (p. 124), shows a revolving screen washed with water from nozzles attached to a pipe passing through the screen, and Plate LX, *B* (p. 124), shows a revolving screen with nozzles at each end for supplying water.

On the 5-foot dredges there are generally provided two 8-inch pumps, which supply about 3,200 gallons of water per minute. Based on a working time of 22 hours a day, this would be equivalent to 284 inches of water. Assuming 3,000 cubic yards dredged a day, the number of gallons of water used per cubic yard would be 1,400. The water consumption stated in cubic yards dredged per inch of water used is approximately 10.6.

On the small Alaska dredges a 10-inch pump works under a head of 45 to 50 feet, furnishing about 3,000 gallons a minute. Based on a 22-hour day and an average of 2,000 cubic yards dredged daily, this figures 266 inches of water, with 7.5 cubic yards to the inch and nearly 2,000 gallons per cubic yard. On the small flume type of dredge, with 2½-cubic-foot buckets, which has become so popular in Alaska, one 14-inch pump under the low head of about 22 feet is used. This delivers about 5,500 gallons per minute and on the basis mentioned above gives 488 inches of water or 4 cubic yards to the inch, and 3,600 gallons per cubic yard. This type of dredge requires more water than the screen dredge on account of the method of washing the material that dumps from the buckets into a straight flume instead of into the hopper supplying the screen.

On the 9-foot flume dredge of the Conrey company there are two 10-inch pumps and one 12-inch and one 14-inch pump, which supply the screen and the sluices with about 12,500 gallons per minute. As the average capacity of the dredge has been 161 cubic yards per hour, 4,800 gallons per cubic yard has seemingly been required, though the actual amount may have been less, as the pumps may not at times have been worked to full capacity.

On the Conrey No. 4 dredge the pumps furnish more than 12,000 gallons a minute, and the capacity of the dredge has averaged 420 cubic yards an hour, or 7 cubic yards a minute, since starting, indicating that 1,700 gallons of water has been pumped per cubic yard dug.

On the Boston & Idaho dredge there are three 14-inch pumps, which, when handling the maximum capacity of gravel, furnish nearly 20,000 gallons of water a minute. As more than 700 cubic yards an hour has at times been dug by this dredge, this dredge will also use approximately 1,700 gallons of water per cubic yard.

ACCIDENT PREVENTION ON GOLD DREDGES.

"Safety-first" rules for the prevention of accidents should be in force on gold dredges as in any well-conducted machine shop or other industrial plant. They will increase the efficiency of the men about the dredge as well as to save them from accidents resulting from their own carelessness.

All belt railings, gear guards and gear railings on dredges and in machine shops should be so arranged as to make it difficult for the men to step between the top and bottom of a running belt or through the slow-running bull wheels. The form of these guards will vary according to the arrangement of the drives and according to the construction of the machinery to be guarded, suiting the climatic conditions under which the dredge is to work.

The general safety orders of the industrial accident commission of California prescribe the following:

All gears, where exposed to contact, must be entirely inclosed, or equipped with side flanges extending beyond the root of the teeth.

All spoke gears and open-web gears which are over 18 inches in diameter, where exposed to contact, must be entirely inclosed. On large gears, such as those in heavy shears or punches, the guard must be such as to cover them to a height of 7 feet above the floor.

Where it is clearly impracticable to cover gears, as described above, a boxed frame of metal or wood must be installed, completely shutting off the machinery gears.

All gear guards must be kept in place while the machinery is in operation.

All belts, ropes, or chains driving machinery or shafting, and all secondary belts, ropes, or chains where exposed to contact, must be guarded. In all cases the point where the belt, rope, or chain runs onto the pulley, sheave, or sprocket, if within 7 feet of the floor or platform, must be guarded.

Exception: Belts which are so small or so slow moving that they are not in any way a source of danger.

All horizontal belts, ropes, or chains driving machinery or shafting, 7 feet or less above the floor or platform, where exposed to contact, must be guarded. All overhead belts 6 inches or more in width and over 7 feet from floor or platform, must be guarded underneath and on sides unless so guarded that persons can not pass under them. All chain or rope drives over 7 feet from the floor or platform must be guarded in like manner to belts over 6 inches in width. In all cases the guard should cover the outer faces of the two pulleys or sheaves and extend upward to such a point, and be attached in such a way that in case the belt, chain, or rope breaks the guard will withstand the whipping force.

Vertical and inclined belts must be guarded substantially as follows: If the guard must be less than 15 inches from the belt, with a complete inclosure of wood or metal to a height of 6 feet above the floor. If the guard can be placed with at least 15 inches clearance from the belt, with a two-rail railing at least 3½ feet high.

Oiling facilities and convenience in repairing should receive special consideration with a view to minimizing the chances of accidents to the machinery itself, as well as to the crew. Except for motor and pump bearings, practically all the bearings on a gold dredge are lubricated with grease, and therefore must be filled either every shift or every day. As there are a rather large number of bearings, the grease cups and grease-gun openings should be placed conveniently so that the oiler may grease as many bearings as possible while the dredge is running, without exposing himself to any danger. If the grease cups are placed some distance from the bearing proper, larger pipes should be used so as to decrease the friction.

The lower tumbler may be greased by means of mechanically operating grease pumps driven from the upper tumbler shaft, and equipped with a pressure gage in the pilot house to indicate that they are working properly. The passage for the grease may be drilled through the whole length of the lower ladder end casting to which it may be connected by a 1¼-inch pipe running under and outside the ladder flange. On the upper end, this pipe is connected by means of a metal house, to the pump. Each side of the lower tumbler is lubricated independently.

The ladder roller bearings have to be greased separately with a grease pump. A considerable saving in weight can be effected by making the barrel of the grease pump of aluminum. The oiler has to walk up and down the inclined ladder on the ladder flange, and he will readily appreciate the difference in weight.

Platforms should be provided on both front and rear gantries for oiling the ladder-hoist sheaves, the spud sheaves, and the stacker sheaves. If a double ladder hanging is used, there should also be a platform for the lower sheave blocks. The platforms should be made strong enough so that they may be utilized in repairing the parts mentioned. All stairways should be provided with smooth railings. Platforms should have both railings and toe boards wherever practicable. All sheave blocks should be equipped with separate oil holes and grease cups for each sheave.

The hubs of operating levers, as well as the shaft bearings for lever connections, should be provided with oil holes or grease cups. Proper use of them will prevent the sticking of levers, caused by the cutting or rusting of shafts and bearings, and will also make the handling of the levers less arduous.

Efficient oiling arrangements not only add to the safety of the oiler, but increase the life of bearings and shafts, and result in better running time.

Drip pans should be arranged underneath the gear trains and bearings to catch excess grease, to help to keep the dredge clean, and to prevent accumulation of grease on the deck and floors.

For bucket-line repairs, the ladder should be provided with cleats or equivalent members, to which cables may be hitched. They should be so arranged that a hitch may easily be made around them.

Keys for connecting the hubs of revolving or swinging members to shafts should, if possible, be of the head type; if this is impracticable, arrangement should be made for backing out the keys. For the flange couplings of direct-connected centrifugal pumps, the Woodruff keys for hubs and shafts seem to be preferable to the ordinary taper keys for insuring easy disconnection of the coupling when ring members of the lamp shaft have to be renewed and are too small to slip over the couplings.

The seats of hubs having a press or shrink fit on a shaft should be made about one-sixteenth inch larger in diameter than the remainder of the shaft, for convenience in backing the gears or pulleys connected to the shaft.

Belt speeds on dredges, especially on the first countershaft on the bucket-drive motor, should be kept below 4,000 feet per minute, as at a higher speed the problem of shaft and pulley balance enters, and bursting of clutches, owing to centrifugal and other forces, is possible.

Electric wiring should, in general, be installed according to the fire underwriters' code. Wires should be run in metal conduits. For a 2,200-volt circuit, lead-covered cables, with all joints carefully soldered, are recommended. Outlets close to the motor terminals should be arranged to guard against mechanical injury. A slot or hole in the conduit at the lowest point of the circuit will prevent accumulation of water from any source. The wires to the controllers of the variable-speed motors should preferably be run in conduits to prevent fouling with operating levers below the floor of the pilot house.

Rubber matting on floors below the switchboard and below motor starters has proved of advantage. Discarded conveyer belt may be used for the purpose. Care should be taken to see that gravel and grit are not allowed to be tramped into the rubber mat, as a small puncture will allow a ground to be made. Mats held together with wooden pins are recommended as best, but they must be kept dry at all times when in use. It is suggested that a wooden mat be placed on top of a rubber matting. To prevent stumbling, the two ends and the side away from the switchboard should be beveled. Rubber gloves are valuable for the crew for handling shore cable and extension pole lines.

As climatic conditions and operating requirements influence the most desirable construction of electric motors no detailed remarks need be made about them except that they should be fully guarded by railing or wire netting.

If transformers are used on board the dredge, they should be housed as independently as possible from the rest of the dredge. On a wooden hull dredge, the transformer house should be lined with asbestos and galvanized-iron sheathing. The switchboards should be equipped with all the safety and protective devices used on any first-class electrical installation.

Flanges on water piping should be made of material to withstand vibration strains set up in digging.

For pump-discharge valves and for all other gate valves used on dredges, outside spindle valves or valves equipped with indicators are desirable, as they give instant information as to the amount of valve opening. Foot valves can be made so that the valve seats are accessible and so that the valve flaps may be replaced without hoisting the suction out of the water.

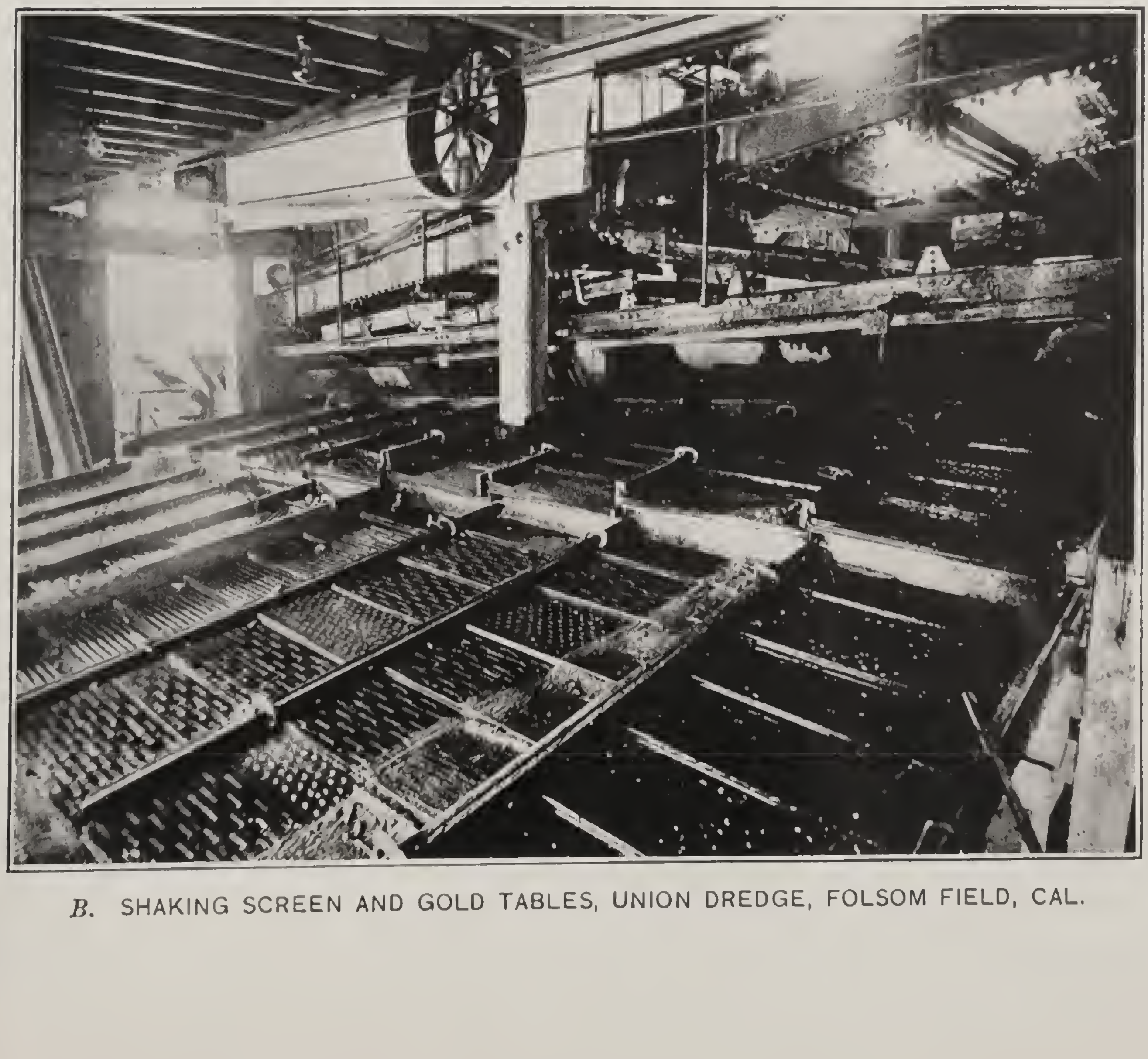
Although the underwriters' rules specify fire pumps on board of dredges, these pumps are generally useless as they are driven by the same power that operates the dredges and the usual cause of fire is through some disarrangement of the electric current, which of course puts all the motors out of commission. It would be better policy to have an auxiliary fire apparatus consisting of a quick-starting gasoline engine which could be started at a moment's notice, attached to a fire pump with 150 feet of hose, and placed on a barge alongside the dredge. This would be a precautionary measure that would appeal to insurance companies and result in a reduction in rates.

In addition to having a number of effective hand extinguishers at readily accessible places, a large chemical extinguisher on wheels with a capacity of 20 gallons or more, with hose connected, should be supplied. The first cost of such an extinguisher is small, and several disastrous fires could have been averted had the dredges been supplied with them.

The cause of the Viloro dredge (Pl. LXI, *A*) being lost by fire was thought to have been a breakdown or short circuit in the main bank of transformers, as flames suddenly burst through the transformer housing. Three men were on shift and two of them must have seen the flames at about the same time and rushed for the fire apparatus. One of the men who was on the lower deck made an effort to start the high-pressure pump that was connected to the fire mains, only to find that the power was off owing to the panel at the substation being thrown out. Thus a first-class pumping plant, attached to a separate set of transformers, proved inadequate in time of need.

The lack of cleanliness and the careless handling of oil and other inflammable material about dredges can not be too severely condemned. It is better to be safe than sorry, and a little attention

A. WRECK OF VILORO DREDGE.



B. SHAKING SCREEN AND GOLD TABLES, UNION DREDGE, FOLSOM FIELD, CAL.

to some of these rules is certainly justified. Oil may be readily removed from floors by sprinkling them with air-slaked lime and then brushing them with a stiff brush.

It is well to have sand within easy reach to use in overcoming electrical disturbances at transformers, and the judicious use of asbestos board or other fireproof material at exposed places is advisable. In addition, on wooden dredges, a few dollars can be advantageously spent in liquid stone or other fireproof paint for exposed woodwork. The new steel dredges are regarded as fireproof, and no fire insurance is carried on them.

The foregoing general suggestions are offered for consideration. The application of such measures is the business of the dredge builder. The instruction of the dredge crew as to their respective duties regarding operation, use, and care of tools and machinery, inspection of hoisting and swing lines, proper anchoring of the swing lines, and the like is part of the dredge master's task.

GOLD-SAVING APPLIANCES.

The gold-saving appliances of a modern dredge consist of a revolving screen to disintegrate and classify the dredged material, a distributor, and a series of tables and sluices fitted with riffles. Formerly the shaking screen had a number of advocates, but all dredges built in recent years have been equipped with the revolving screen which has proved its superiority as a disintegrator and classifier, especially when the material is at all difficult to wash.

With a few exceptions, the metallurgy of dredging is simple, and until recently few changes have been made in the gold-saving arrangements of a dredge during the last six or eight years. Although considerable attention has been given to increasing the yardage handled by the dredge and to increasing the operating time, it was thought that additional table area was sufficient for all metallurgical requirements. In the early days of dredging experiments were made to ascertain the loss of gold in dredge tailings, but it was difficult to obtain a representative sample. Although most of the tests showed a loss of some gold, the loss on the best managed dredges was generally estimated to be not more than 5 per cent of the total content for the ground handled, and many of the operators were content to let well enough alone. Present-day tests have demonstrated that there has

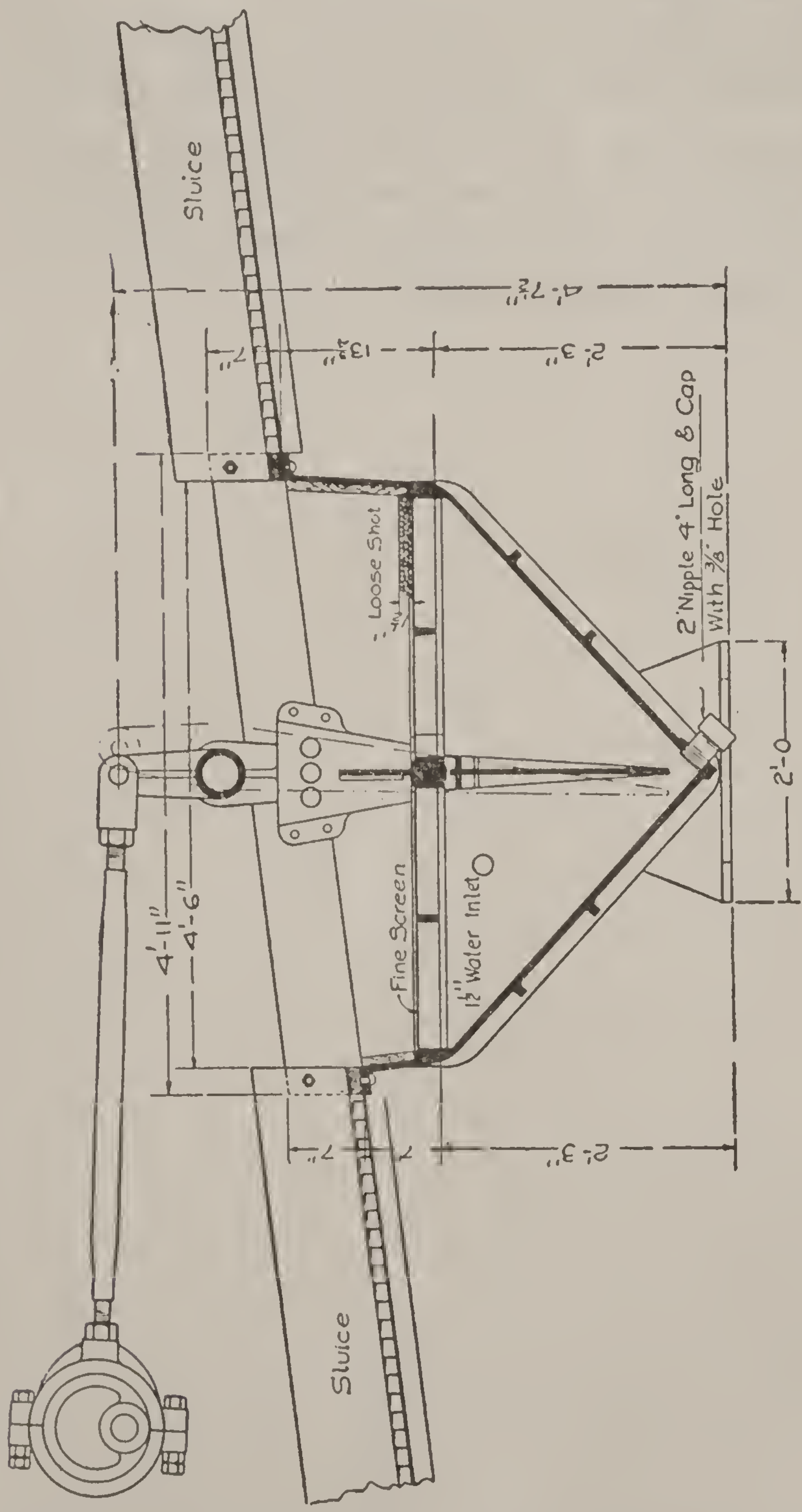


FIGURE 19.—Neill jig and eccentric drive.

been a considerable gold loss at times, and far more attention is being given to improving the percentage of gold recovery. The most important change that has been made to this end is the use of jigs to replace or supplement tables, and the grinding of jig concentrates in order to brighten the rusty gold so that it will amalgamate.

USE OF JIGS.

According to the author's recollection, the use of jigs on the gold dredge was first suggested by Robert H. Richards,[a] following experiments made by the United States Geological Survey at Portland, Oreg. It remained for J. W. Neill to first try out this idea on a working scale. Mr. Neill describes[b] his tests with the jigs on the Yosemite dredge in California. (See figs. 19, 20, and 21.) Subsequent experiments with one of these jigs were made on Dredge No. 7 of the Natomas Consolidated Co. It was found that a considerable quantity of gold failed to amalgamate and was lost in the tailing. A number of changes and improvements were made in the jig, and it has to-day proved a decided advantage in saving fine gold and gold that will not easily amalgamate.

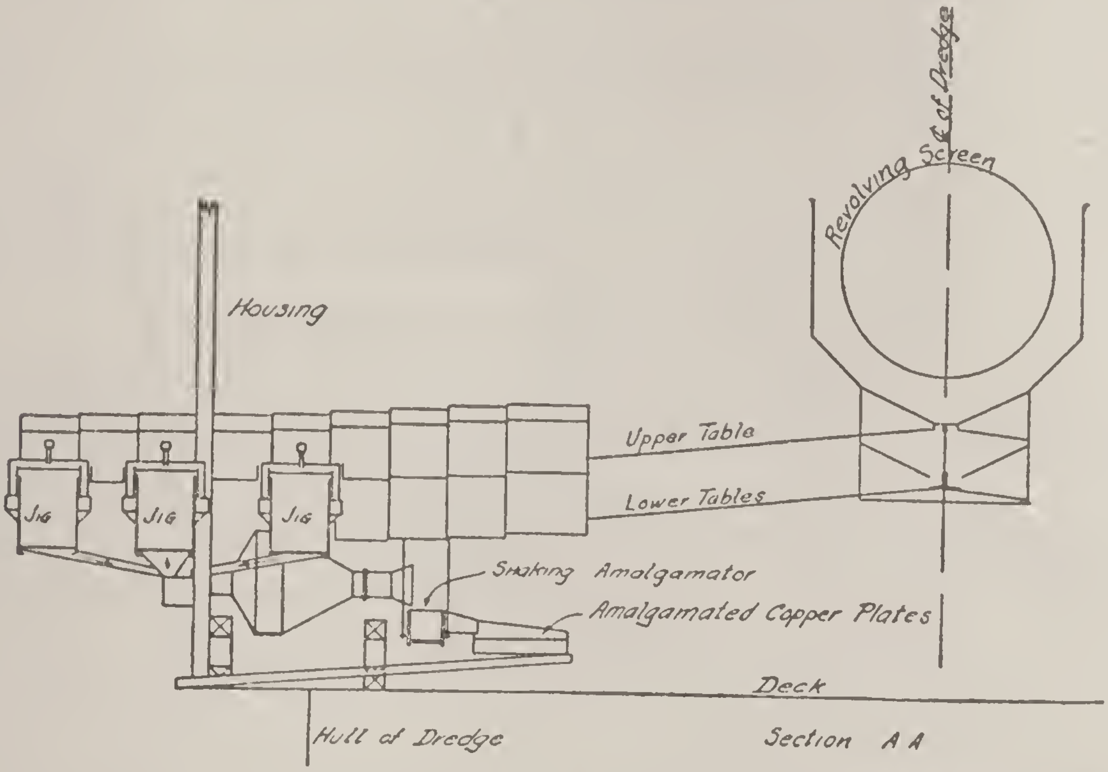


FIGURE 20.—Cross section showing arrangement of jigs, Hardinge mill, and amalgamator, Yosemite dredge.

GOLD-SAVING EQUIPMENT ON TWO NATOMA DREDGES.

The installation on the No. 7 dredge consists of 8 jigs and two 4½-foot Hardinge pebble mills. Four of the jigs and one of the mills were placed on each side of the dredge. On the No. 10 dredge 10 jigs are used—5 on each side. The jigs are placed in the first four sluices of the gold tables leading from the upper, or forward, end of the screen, and the mills are placed on the main deck, so that the concentrates from the jigs will run by gravity to the feed box of the mill.

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[a] Richards, R. H., Utilization of black sand: Eng. and Min. Jour., vol. 83, June 29, 1907, pp. 1251-1252.

[b] Neill, J. W., Application of jigs to gold dredging: Min. and Sci. Press, vol. 109, Nov. 28, 1914, pp. 839-842.

The results of several tests made at different times indicate that the two mills handle approximately 75 to 80 tons of concentrates a day, on the basis of a daily average running time of 20 hours. The sluices in which the jigs are placed recover approximately 75 or 80 per cent of the gold that is recovered on the dredge.

The value of the concentrates from the jigs varies from 75 cents to about $1.50 a ton, from about 6 to 12 per cent of the gold cleaned up on the tables. Tests made for comparative purposes indicate that not less than 75 per cent of the gold passing through the mill is recovered by the jigs and mills.

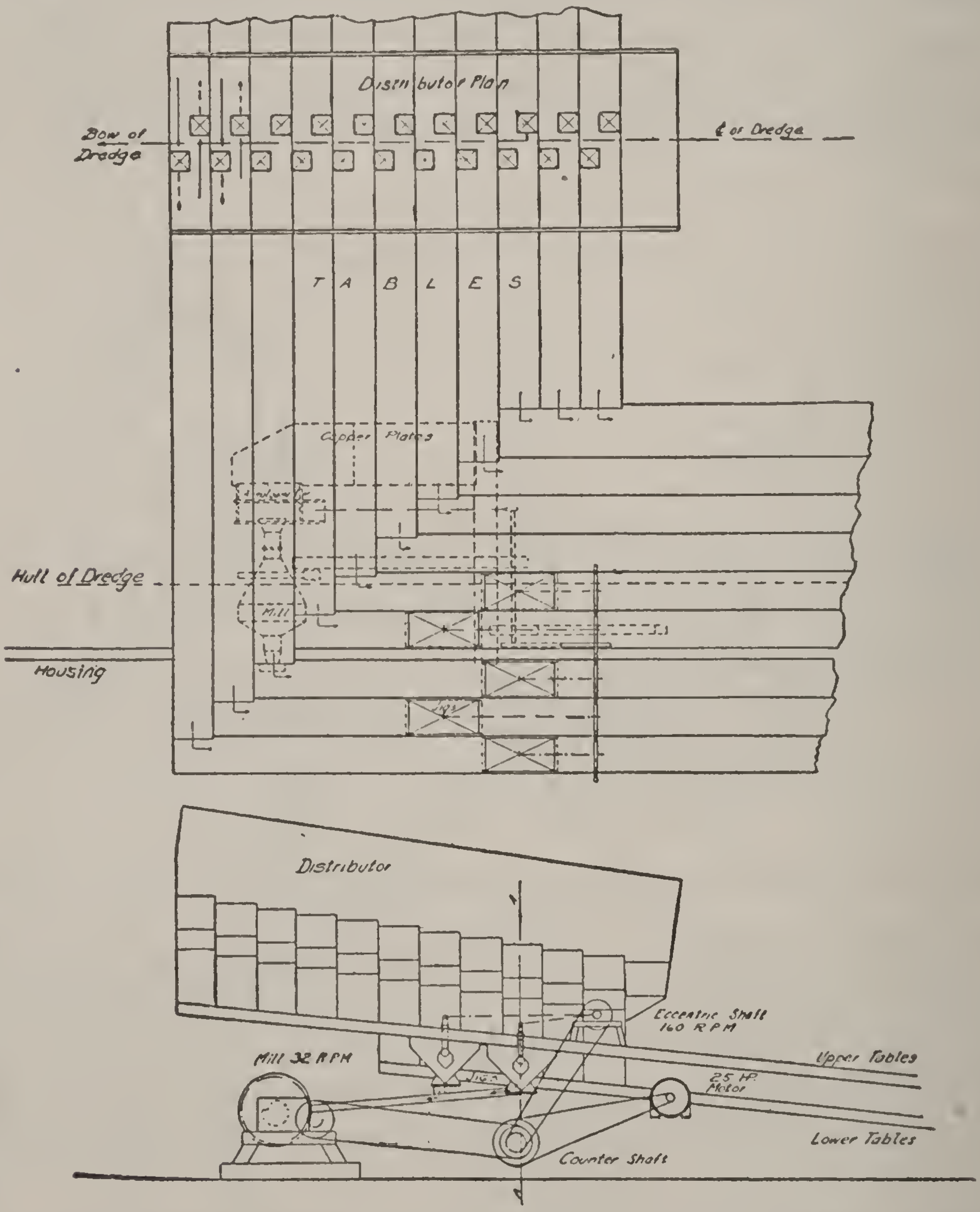

FIGURE 21.—Plan and elevation of tables on Yosemite dredge, showing position and drive of jigs and mill.

Several tests to determine the saving of gold after it had passed through the mill showed that the losses between the heads or concentrates from the jigs and the tails, after the material had passed through the mill and over the plates, were from traces to 5 cents per ton. The valuable material lost was in the form of minute particles of gold amalgam and floured quicksilver.

The following figures regarding the Natoma dredges were furnished by Griffin:[a]

Each bank of jigs and one mill are driven by a 25-horsepower motor. The total power consumed is about 36 horsepower, divided about 18 horsepower to the mills and 18 horsepower to jigs.

The labor cost averages about $110 a month; material, about $25; and power, about $120; total, $255.

We have jig and mill equipment on dredges Nos. 7, 8, 9 and 10 and a jig without mill on No. 4. My data are complete on only dredges Nos. 7 and 10, results from which follow:

Gold recovery from mills and jigs of Natoma Nos. 7 and 10 dredges.

Dredge No.	Period days.	Cubic yards dredged.	Gross clean-up.	Value per cubic yard.	Mill and jig clean-up.	Value per cubic yard.	Mill and jig clean-up, per cent of total.
				Cents.		*Cents.*	
7	176	842,014	$108,153	12.84	$12,879	1.52	11.90
10	256	1,973,661	215,444	10.92	15,736	.80	7.30

The recovery percentage from Nos. 8 and 9 is somewhat lower than from No. 10 and will average about 5½. The recovery on No. 4 is so low that I am considering removing the jigs unless there is a change for the better in the near future.

CHARACTER OF GOLD-SAVING TABLES.

Gold-saving tables are made either in single or double banks. Single-bank tables are used on dredges up to 7½ feet, but in the larger boats the riffle area obtainable on a single set of tables has generally been found insufficient, and to meet the requirements of the increased yardage a second set of tables was arranged, thus doubling the riffle area. The double-deck table was developed in 1908, and the use of four tail sluices of different lengths made it possible to distribute the gravel over a greater area and to discard the sand pump which was always a source of trouble.

The single-bank tables are usually made of wood and the double-bank tables of steel. The tables are supplied with some form of the so-called Hungarian riffles, either of wood shod with an iron bar on top, or of angle iron, or of cast bars of manganese or other steel. The angle-iron shape is still considered by many as the most effective, for the water passing over the riffles forms an eddy under the projecting leg which facilitates catching the gold. Amalgam or quicksilver riffles are set in each sluice and the quantity of quicksilver used has greatly increased, on some dredges being about ten times the amount formerly used.

CONSUMPTION OF QUICKSILVER.

The consumption of quicksilver on several dredges under Oroville conditions is shown by the accompanying table furnished by Mr. C. G. Leeson, of Oroville, and is of interest in being, so far as the author knows, the first table giving such results that has been published.

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[a] Griffin, F. W., personal communication.

Quicksilver losses on Oroville dredges.

Period.	Name of dredge.														
	L. B. No. 2.			L. B. No. 3.			Empire.			Hunter.			Victor.		
	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.	Number of clean ups.	Amount of loss per clean up.	Per cent lost.
		Ounces.			*Ounces.*			*Ounces.*			*Ounces.*			*Ounces.*	
First six months of 1908	No record.														
Second six months of 1908	9	65.6	1.9	11	26.7	1.07	12	47.1	1.3	12	221	3.6	9	67	2.4
First six months of 1909	12	114	3.5	12	69.7	2.5	12	47.2	1.4	12	52	1.1	12	83	2.8
Second six months of 1909	12	92	3.1	12	144	5.4	7	108	3.6	12	61	1.5	11	49	3.03
First six months of 1910	No record.														
Second six months of 1910	13	98.5	2.6	9	192	2.3	9	67	1.2	7	74	1.05	8	106	2.04
First six months of 1911	No record.														
Second six months of 1911	12	93	3.35				11	85	2.66	12	211	6.64	12	158	5.48
First six months of 1912	12	102.8	3.75				12	85.9	2.6	12	86	3.07	11	117	4.5
Second six months of 1912	13	97.8	4.1				13	83.5	2.97	11	190	8.0	12	141.5	4.7
First six months of 1913	12	88.25	4.14				11	99	3.06	10	126.5	3.15	12	69.58	2.26
Second six months of 1913	12	116	5.06				13	65	2.04	12	54	1.56	12	81	2.60
First six months of 1914	11	96	4.12				11	115	4.07	12	82	2.35	12	91	2.91
Second six months of 1914										13	120	3.2	12	55	1.55
First six months of 1915										12	144	3.5	11	49.5	1.5
Second six months of 1915															

NOTE.—The average per cent lost on the L. B. No. 2 per 6-month period was 3.5 per cent; the average per cent lost on the L. B. No. 3 per 6-month period was 2.72 per cent; the average per cent lost on the Empire per 6-month period was 2.46 per cent.

The quantity of quicksilver placed on the dredge tables varies with the size of the dredge and with the ideas of the operator—from 150 pounds or less on the smaller dredges up to 3,000 pounds on the larger dredges. The Yuba Consolidated Co. has gradually increased the quantity of quicksilver placed on the tables and found a corresponding improvement in the percentage of recovery. A few years ago 150 pounds was considered sufficient for the 7-foot dredges, but now these dredges with a table area of 2,800 square feet use about 1,000 pounds for the sluices and traps.

On some small dredges the loss of quicksilver has amounted to as much as 7 pounds a week when only 2 flasks at a time were being used on the tables. This would be about 5 per cent of the total amount used. On the Jenny Lind dredge the loss of quicksilver was greatly reduced when the clay sluice previously described (p. 120) was put in, showing how the clay had robbed the sluices.

FACTORS AFFECTING EFFICIENCY OF TABLES.

The efficiency of the gold-saving tables (Pl. LXI, *B*, p. 136) is affected by the grade. It is generally found that a grade of 1½ inches to the foot is the most efficient slope for the tables and sluices, though the nugget sluice of the Tonapah No. 3 dredge described elsewhere is run on a steeper grade, and at Ruby, Mont., a grade of 1 inch to the foot is used.

Another factor affecting the efficiency of gold-saving tables is the quantity of material sent to the tables. If the grade of the sluice is too flat, or the volume of water insufficient, there will be an accumulation of sand, and amalgamation will be hindered. If the tables are crowded beyond their capacity there will be a loss of gold-saving efficiency.

A third factor affecting gold-saving tables is the amount of water. Too much water may carry off some of the fine gold, but more gold is lost from insufficient washing than as gold carried in suspension. The proper amount of water to use in dredging varies with the material dug, but the principle to follow is to regulate the water so as to bring as much of the gold as possible into contact with the quicksilver.

At Ruby, Mont., in order to test the efficiency of the tables, additional gold-saving devices consisting of undercurrents were placed on Dredges Nos. 3 and 4. The undercurrents on the No. 3 dredge, a tail-sluice dredge, are placed on the tail-sluice pontoon, and consist of 10 tables connected by launders to the lower end of the tail sluice, where the fines pass through a grizzly. On the Conrey No. 4 dredge, the undercurrents are placed near the stern immediately under the main gold-saving tables. There are 12 tail sluices to the tables on this dredge, and at the lower end of each tail sluice there is a plate perforated with holes three-eighths of an inch in diameter, through which the fines pass onto the undercurrent tables. The gold saving tables and sluices of this dredge are shown in Plate LIII, *B* (p. 98).

During 36½ months' operating the return from the undercurrent arrangement on the No. 4 dredge amounted to about 0.5 per cent of the total recovery, indicating the efficiency of the tables and sluices of the stacker dredge when very fine or rusty gold is not present. On the No. 3 sluice dredge the undercurrent tables and undercurrents have shown recoveries of something like 8,000 ounces of amalgam since December, 1908, indicating that there is greater danger of loss of fine gold on dredges of the sluice type than on dredges of the stacker type.

The recoveries made by the save-alls on the different dredges are indicative of possible losses from spilling from the buckets caused by stickiness of material and overfilling and noncleaning of buckets. The different gold-saving devices in the well save-alls are somewhat similar; but on some dredges it is easier to guard against a spill than on others. On the Conrey No. 4 dredge it is possible that there is less spill than on any of the other Conrey dredges and consequently less percentage of gold caught. The recovery from the well save-all varies from 2.76 per cent of the total clean-up on the No. 4 dredge to 5.01 per cent on the No. 3. The undercurrent system was used on the No. 4 dredge mainly with the idea of determining the loss of gold by the tables, and it has proven a comprehensive check on this work besides resulting in a saving of over 1,000 ounces of amalgam to July 1, 1915.

The nature of the gold recovered from the Conrey ground is distinctly coarse, but not many nuggets are found. Even in the No. 3 sluices the largest nugget so far recovered was worth $17 and there are few nuggets found that are worth more than $5. The fineness of the gold varies from 0.836 to 0.873, the lower-grade and coarser gold being found in the upper part of the gulch.

At Breckenridge, Colo., the screen of the No. 3 dredge of the Tonapah Placer Co. is filled with plates next to the lower section, having 2½-inch holes. The fines from the tail of the screen go to the "nugget" sluices, built somewhat steeper than the main tables. This arrangement results in the recovery of considerable nugget gold that would otherwise be lost.

Undercurrents were put on the No. 3 sluice dredge of the Tonopah Placer Co. at Breckinridge, and resulted in large savings of gold formerly lost. A sketch of the undercurrent system, furnished by Mr. E. J. Hopkins, is shown in figure 22.

Undercurrents have not been extensively tried in California except on dredges of the sluice type; on one California table dredge a blanket was tried on an undercurrent in the main sluice, but the recovery made was not of much importance. The placing of undercurrents on the main sluices of some dredges would unquestionably effect a saving even greater than that made by the undercurrent on the No. 4 dredge at Ruby, and the practice should be more widely adopted.

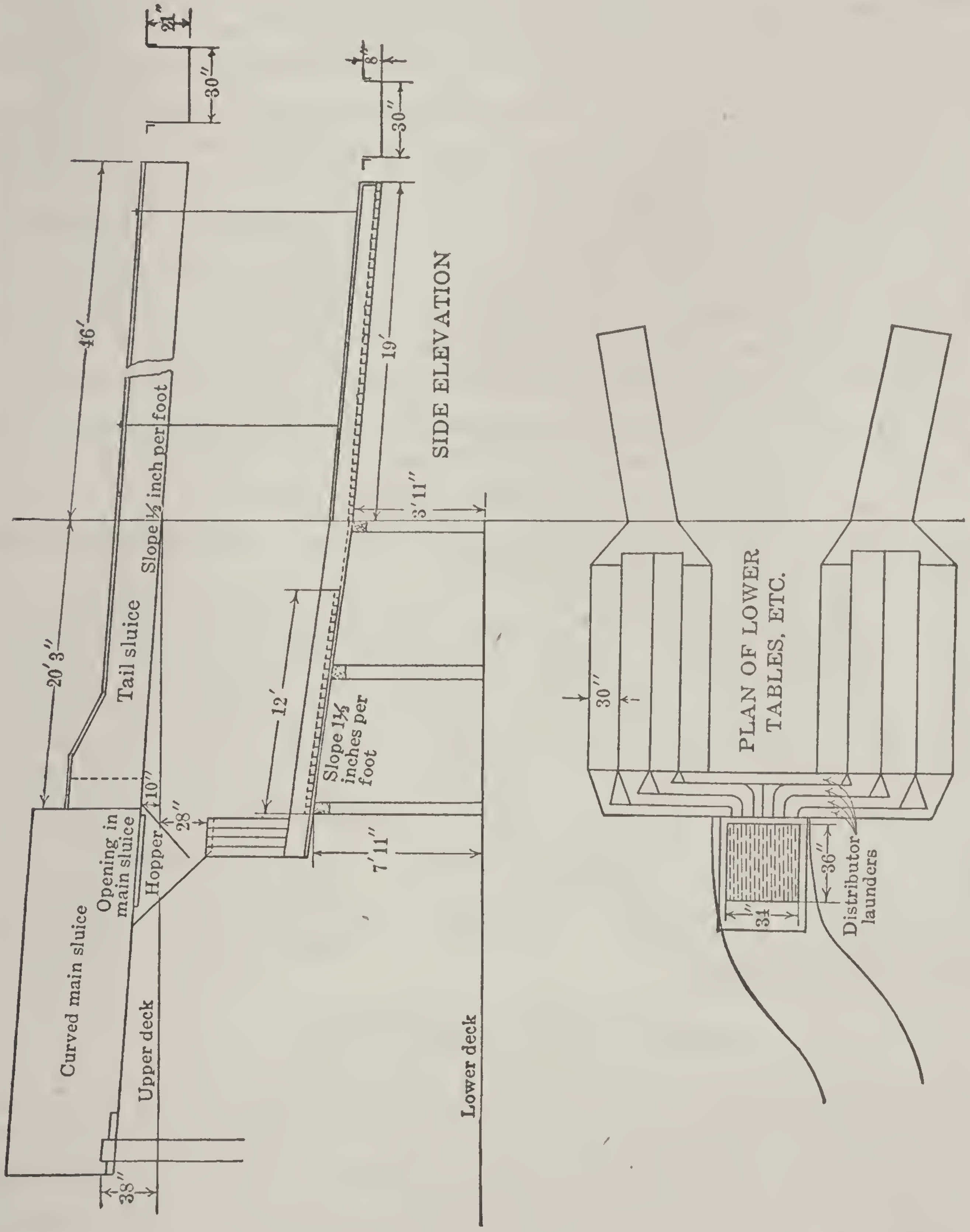

FIGURE 22.—Plan and elevation of undercurrent system used on No. 3 dredge at Breckenridge, Colo.

TABLE AREA.

The table area varies on different makes of dredges and with the ideas of different operators, although it is governed to some degree by the character of the gold in the ground handled. A good rule for the table area seems to be to make it as large as possible, as a safeguard against losses.

California practice on the large dredges handling 350 to 500 yards an hour is to have a table area of 6,000 to 8,000 square feet. Dredges handling 140 to 300 yards an hour have 2,000 to 2,800 square feet of table surface, and smaller dredges 1,000 to 1,500 square feet. These figures may be compared with those resulting from the practice at Ruby where, as saving of the gold is easy, less than half of this space is used. The Ruby figures are as follows: The No. 4 dredge, handling an average of 420 yards an hour since it started operation, has 2,955 square feet. Dredges Nos. 1 and 2, handling averages of 174 and 132 yards per hour, have 1,200 square feet of table area, and the sluice dredge, which has handled an average of 161 yards an hour since commencing operation, is equipped with approximately 1,000 square feet. On the 18-foot dredge of the Boston & Idaho Co., which handled more than 4,600,000 cubic yards during 1914 and more than 600 yards an hour at times, the table area is approximately 4,000 square feet and a part of this table area was cut out as being unnecessary, most of the gold being coarse and easily saved, though there may have been some loss of fine gold. The dredge of the future will no doubt be equipped with jigs to take the place of a considerable part of the table area, and experiments with jigs will be watched with interest.

Water is supplied to the revolving screen by a high-pressure pump through nozzles placed at either end or both ends of the screen (Pl. LX, *B*, p. 124), or by a pipe passing through the center of the screen which is equipped with adjustable nozzles (Pl. LIX, *B*, p. 124). Additional water is fed the tables by the low-pressure pumps. The grade of the screen is about the same as that of the table. It is vital that plenty of water under sufficient pressure be fed to the screen in order to insure satisfactory disintegration and washing of the material, especially if clay is present. A high loss of gold is due to inefficient washing in the screen.

LOSS OF GOLD IN SUSPENSION.

Although the greatest percentage of the gold recovered in dredging is caught on the upper parts of the first three or four tables, doubtless there is on some dredges a considerable loss of gold caused either from insufficient washing of the material on the screen or too small a quantity of water supplied the tables, and on most dredges some quicksilver will be found at the lower end of the tail sluices. In addition a considerable amount of gold is carried in suspension in dredge water. Tests to determine whether there was any loss have been made on several fields with the same result. Samples were taken of the water while the dredge was in operation and of the mud that settled in the dredge pond. The latter test showed that the gold was lighter than the mud and settled more slowly, and the farther the sample was taken from the stern of the dredge the richer it proved to be after the water had been evaporated.

No one has yet worked out a scheme entirely to overcome loss; some very fine gold is caught in the Neill jig and the Hardinge mill arrangement put on the Natoma No. 7 dredge; the concentrate from the jig proves on examination to contain not only rusty flake gold, but also very fine and floury bright gold, but it is doubtful whether this fine gold was previously being carried in suspension. Much of the gold that is carried in suspension would amalgamate if brought into sufficient contact with the quicksilver, but on account of its finely divided condition it is carried over the tables in suspension with the water and is one of the gold losses in the present method of handling the dredge material. This loss seems largely unavoidable. The concentration necessary for the saving of the main gold content causes a loss of some of this light material and in addition to the loss there is some gold that is put in suspension by the breaking up of the ground by the bucket line and does not go over the tables at all.

CLEANING OF TABLES.

The gold tables are cleaned at regular intervals, varying somewhat with the character and the gold content of the material. On the large dredges the main tables are cleaned every week and the side sluices every other week. For doing this work the large dredging companies employ clean-up crews, the dredge crew assisting as far as necessary, or attending to overhauling and repairing the dredge.

The time required to clean the table and sluices varies from 4 to 8 hours on the different dredges. The method of cleaning varies, but the principle is the same. Commencing at the upper end, the riffle frames are taken up, washed, and laid to one side. Water from a small hose under pressure is used to assist in gathering the amalgam and the concentrates, and these are scraped into buckets and washed in a "Long Tom" or small sluice. The amalgam and quicksilver is collected and strained and the amalgam retorted and melted.

In some districts shot, lead, nails, coins, etc., are picked up by the dredge and lodge on the tables. This material, all of which collects some amalgam, forms a second product which is treated whenever a sufficient amount accumulates. The following method of cleaning was developed at Ruby, Mont.:

Above the last stop in the lower end of the table is fitted a converging iron trough, emptying through a short length of pipe into the clean-up box. The riffles are taken off the table, water is turned on, and all the gravel, gold, mercury, etc., is washed into the clean-up box, which is 2 feet 11 inches long, 18 inches wide, and 15 inches high. The box is divided into four compartments by an iron plate extending from the top of the box to within about 3 inches of the bottom. In each of the two central compartments is an iron plate 3½ inches high, extending from the bottom. The material entering the box must pass alternately under and over the partitions to get to the overflow of the box. The box is placed on a slope of 1½ inches in 12 inches, and mercury is placed in front of each of the bottom plates and at the lower side of box, thus forming three mercury traps through which the gold and sand must pass. The material overflows from the box into a small tail sluice containing matting and expanded metal and thence down the tail sluice of the dredge. This method has been demonstrated to be most efficient. Any gold not fully amalgamated can be worked into amalgam by slightly stirring the box as the material passes through. Two men clean up the Conrey No. 4 dredge, one of the largest in the world, in four to five hours, saving wages and time as well as making a more perfect clean-up than by the former method employed.

GOLD SAVING IN YUBA FIELD.

The following notes on gold saving in the Yuba gold field represent the practice followed by one of the world's leading companies. Over $20,000,000 has been taken out to date. Owing to excellent natural conditions gold saving in this field is rather simple. The ground is clay, gravel, and sand, with practically no loam and little clay except on bedrock. There is plenty of clean water, so that screening and washing are easily handled.

The dirt goes from the buckets to the hopper, the buckets being washed with a heavy stream of water to insure clean dumping. What dirt does hang to the buckets falls on the grizzly above the bucket idler. The fines go through and are washed over riffles in the save-all, the construction of which is similar to that of the main gold tables.

Water is supplied to the material in the screen through nozzles in the upper end and in the lower end of the screen. The washing is effective and the gravel that goes to the conveyer belt is clean. The undersize from the screen pass through the distributor to the gold tables, where more water is fed from other pumps. On account of the high proportion of sand in this ground, more water is necessary than under usual conditions, a most important consideration.

On the large dredges the gold tables are in two banks, 10 sluices in each, on both sides of the screen. The sluices are 30 inches wide and are set with a fall of 1½ feet to 12 feet. The total table area is 7,800 square feet. This may be compared with the table area of the No. 4 dredge at Ruby, 2,900 square feet, as showing the different requirements in handling fine and coarse gold.

The ordinary Hungarian riffle used here is made of wood with an iron bar on top; and some experiments are being made with manganese bars for this purpose.

A number of trap or quicksilver riffles are set in each sluice. These are made much like the ordinary riffle, except that the bottom is solid and the spaces between are divided into compartments which are filled with quicksilver. These riffles have been found to be efficient The quicksilver is apt to sink down in the soft sand in the other riffles and not come in contact with the dirt. About 3,000 pounds of quicksilver is used on the tables every week on the No. 14 dredge. The 7-foot boats with a table area of 2,800 square feet use about 1,000 pounds a week. This increased use of quicksilver has been one of the important changes. A few years ago 150 pounds was considered ample for a 7-foot boat.

The clean-up is made every week. The riffles are taken up and washed. The dirt is concentrated at the stops to about 20 per cent. of its original volume, much light sand being left with the amalgam, quicksilver, and black sand. This light sand often carries some rusty gold. The concentrate is taken up and washed in a "long Tom," and the amalgam, quicksilver, and most of the platinum are taken out. The sand is screened at the lower end through an 8-mesh screen and the fines are sacked and taken in to the mill.

The mill is an amalgam barrel with a long flat sluice beneath. The sand is added in the following proportion: One thousand pounds to 20 pounds of mercury, $\frac{1}{4}$ pound of cyanide, and 80 gallons of water. This mixture is ground two hours and washed out in the sluice. The cyanide is an important factor. It keeps the quicksilver in good shape and aids in amalgamation. There is a gain of about 10 pounds of quicksilver to 100 tons of sand if cyanide is used, and a loss of about 20 pounds if it is not used. It almost entirely prevents flouring of the quicksilver and brings together the flour gold that is in the sand.

Tests made of the sand from the mill after washing have showed practically no gold or platinum.

The retorting and melting is by the ordinary process.

On most dredges the main gold tables are inclosed with heavy wire netting to prevent pilfering of the amalgam.

WORKING COSTS.

The working costs of gold dredges are always of interest, although it should be recognized that comparisons that are at all representative are difficult. In this section are given a number of working costs for dredges operating in different fields, with notes of some points that should be taken into consideration when making comparisons between different dredges. The experienced engineer, of course, will not fall into the error often made by the inexperienced or casual investigator of trying to compare and reconcile the working results of dredges operating under widely different physical and economic conditions.

REQUIREMENTS IN COST ACCOUNTING.

In making comparisons of operating costs of dredges working in different fields, it is well to keep in mind several important facts. The dredges compared should be operating under somewhat similar conditions, and if transportation and other economic facilities are not closely similar, allowances should be made for any vital difference. Operating costs should be similarly tabulated, for seeming low operating costs often result from a particular method of bookkeeping.

Depreciation charges are generally arbitrary assessments based on previous experience, and occasionally these are left out of stated operating costs. In the matter of repairs, bucket-line renewals are not always charged to working costs though monthly repairs are included. This item alone might amount to ½ cent to 1½ cents a cubic yard.

Figures should be obtained over as long a period as possible, for dredges operating under favorable circumstances and for short periods have shown bare working costs of less than 1½ cents a cubic yard, or approximately 1 cent a ton of material handled. Dredges operating even in the same locality may over different periods encounter entirely different physical conditions in the ground to be dug, as shown by examples given (p. 107). The age of the dredge may also have a considerable bearing on costs, the older dredges generally having higher repair costs.

Similar methods of calculating yardage should be employed, and in this connection the following comparison of yardage dug by the different dredges at Ruby during 1914, as determined by bank measurement and by the theoretical capacity by arranging an accurate automatic tally of the buckets dumping during the same time, is of interest; it illustrates some of the reasons for the seemingly low operating cost per cubic yard occasionally attributed to some of the dredges built and operated outside this country.

Comparison of yardage dug by different dredges at Ruby in 1914.

Dredge No.	Size of buckets.	Average depth dug.	Actual yardage.	Theoretical capacity.	Buckets per minute.	Kilowatts per yard.	Yards per hour.
	Cubic feet.	*Feet.*	*Cubic yards.*	*Cubic yards.*			
1	7	26.6	1,100,906	6,293,400	15.71	1.78	165
2	7	26.2	1,140,195	5,925,300	14.79	1.42	171
3	9	34.6	1,068,164	4,542,590	13.85	1.79	171
4	16	49.7	3,093,513	7,525,224	20.24	1.23	499

In foreign fields the yardage handled is often based on a theoretical capacity of the dredge for the total running time, after some allowance has been made for buckets partly filled. How little this percentage would represent actual conditions is shown by the foregoing table, for with the exception of the No. 4 dredge, which handled 41 per cent of the theoretical capacity, the other dredges averaged only 20 per cent of the buckets actually dumping. In estimating yardage from bucket capacity, it must also be remembered that calculations are generally made for loose gravel instead of from bank measurement, a difference of about 25 per cent.

Similar figures are shown by the following tabulated statement of the dredges of the Natomas Consolidated for the year 1913, the actual capacity being 44.5 per cent of the theoretical for the time in commission, and 55 per cent for the total dredging time.

Record of dredges of the Natomas Consolidated of California for the year 1913.

[Comparison of actual with theoretical capacity.]

Dredge No.	Bucket size.	Depth worked.	Actual total capacity.	Working efficiency.	Actual as compared to theoretical capacity.		Running expense per cubic yard.	Total expense per cubic yard.
					Total time.	Dredging time.		
	Cubic feet.	*Feet.*	*Cubic yards.*	*Per cent.*	*Per cent.*	*Per cent.*	*Cents.*	*Cents.*
Natomas 1	13½	22	2,575,160	78.6	44.8	57.0	1.65	4.09
Natomas 2	8½	27	2,141,869	86.2	59.0	68.4	1.80	4.19
Natomas 3	8½	28	1,726,206	84.9	47.7	56.1	2.17	4.74
Natomas 4	13½	22	2,279,016	79.8	39.6	49.7	1.46	3.56
Natomas 5	9	53	1,048,519	80.3	50.3	62.7	2.03	11.05
Natomas 6	9	48	1,339,077	67.1	38.4	57.3	2.34	8.70
Natomas 7	9	55	1,156,930	80.5	47.6	59.1	2.39	4.65
Natomas 8	15	62	1,807,235	80.6	33.0	41.0	2.87	6.54
Natomas 9	15	64	2,363,888	84.0	40.7	48.4	2.44	6.69
Natomas 10	15	45	2,470,085	78.8	42.5	54.0	2.21	5.15
Feather 1	7½	26	929,308	83.4	32.0	38.4	2.59	6.47
Feather 2	7½	33	1,483,404	84.9	51.1	60.2	1.48	4.15
Feather 3	13½	27	3,090,081	82.2	59.1	72.0	1.82	4.46
Total or average	....	....	24,410,778	80.9	44.5	55.1	2.04	5.35

The following statement from the report of the Tongkah Harbour Tin Dredging Co., Siam, for the year ended September, 1914, clearly illustrates the contentions of the writer and is expressive of the results obtained when different methods of computing yardage are employed:

The working costs were 5.740d. (11.48 cents) per yard, as against 3.186d. (7.632 cents) last year. The smaller yardage output of this year as compared to that of previous years results from a change in the method of calculation. Formerly the yardage was estimated as the product of the number of hours worked multiplied by a yardage factor. Apparently the factor remained the same irrespective of the nature of the ground dredged. Now the yardage is computed from measured width of cut by distance pulled ahead and depth of face worked.

Yardage based on:

Theoretical capacity is based on buckets dumping per minute as follows: Nos. 1, 2, 3, 4, 7—22; all others, 20 per minute.

Efficiency equals total dredging time divided by time actually in commission during the year (less holidays).

Running expenses consist of labor, material, power, and water, and total expense includes repairs, which averaged $2.49, and taxes, insurance, general and other charges, which averaged $0.82 a cubic yard for all dredges.

When yardage costs are based on calculations of theoretical capacity, the advantage would lie with the operator having the most vivid imagination regarding dredge capacity; his operating costs per cubic yard would seemingly be reduced to a most desirable figure, though his total costs would not be affected.

The capacity of a dredge and the working costs per cubic yard do not as a general thing concern the individual operator or the shareholder, and it makes little difference to either how the figures are obtained. What does concern both is that the recovery exceeds the operating costs and leaves a profit. Only when it is desired to make some comparison of the operating cost and the capacity of different dredges do the cost per cubic yard and the method of arriving at the yardage handled become of importance.

Hence working costs can not be fairly used in such comparisons unless uniform and proper methods of determining them are employed and also unless operating conditions are somewhat similar. It is not generally recognized that ground in the same locality often varies so much that dredges similar in construction, design, and bucket capacity show different operating costs per cubic yard; neither is it generally recognized that figures for the same dredge may vary greatly in different years. A case in point is that of the Hunter dredge, a 5-foot close-connected boat of the Oro Light & Power Co., at Oroville. This dredge during operations extending over three years handled an average of about 200 cubic yards an hour. Afterwards the boat was moved to another part of the same property, where for a number of months, on account of difficult digging, it handled only 80 cubic yards an hour. Another example which may be quoted to illustrate different results of similar dredges is the following extract from an old report of Oroville Dredging (Ltd.):

> The 5-foot dredge of the Oroville Dredging & Exploration Co., handling a loose gravel which requires little power to displace, and with all of the conditions as nearly ideal from a dredging standpoint as could be wished, averaged for seven months 2,737 yards a day. Comparing this with the daily average of 1,723 yards obtained by

Dredge No. 2 of the Boston & California, a dredge having the same digging power and (theoretical) capacity, we have a striking illustration of the difference to be obtained in different formations in the same district.

The amount of ground handled by another Oroville dredge with 6-foot buckets also varied in different parts of the property. In some places a cemented overburden had to be broken with explosives; in others much heavy clay, difficult to dig, was encountered. The operating costs and yardage handled may vary considerably over different years, as shown in the following table:

Operating cost of Oroville dredge with 6-foot buckets.

Year.	Cubic yards.	Total operating costs.	Cost per cubic yard, cents.
1910 [a]	649,639	$36,764	5.66
1911	874,845	53,306	6.09
1912	710,257	50,150	7.06
1913	832,970	48,322	5.80
1914	819,721	50,587	7.03
	3,887,432	239,129	6.15

[a] 10 months.

In this connection the following statement is of interest regarding the different costs of Conrey No. 3 dredge: From April to July, 1912, and from August to November, 1913, inclusive, the dredge worked downstream and averaged 163.4 cubic yards a working hour, with an average operating cost of 5.40 cents a cubic yard. During the last four months of 1912 and several months of 1913 the dredge dug upstream and averaged 203.5 cubic yards an hour, at a cost of 4.87 cents, there being an increase of 25 per cent in yardage and a decrease of 10 per cent in cost in working upstream as compared with working downstream.

COSTS FOR SAME DREDGE MAY VARY BY YEARS.

Cost for different years at the same dredge may also vary greatly on account of the method frequently employed of charging all renewals and repairs to the year in which they are made even though the benefit from such work extends over several years. For instance, Exploration No. 3 dredge, of Oroville Dredging (Ltd.), handled 1,228,110 yards during 1912, at a cost of 4.18 cents a yard. According to the annual report of the company the cost of operation was reduced 1.54 cents a cubic yard below that of the previous year; this difference was due to the fact that the bucket line had been overhauled and charged for in 1911, and required little attention during 1912.

Another example of this practice is shown in the annual report of Natomas Consolidated for 1913. The cost of operation of Natoma

Nos. 5 and 6 for 1913 is given at 11.05 cents and 8.70 cents a cubic yard. This is due to the fact that extraordinary repair and reconstruction cost of 8.18 cents per yard for No. 5 and 5.34 cents per yard for No. 6 are charged against the operating costs of the dredges for the year; No. 5 being out of commission 4 months and No. 6 a little over 2 months while undergoing general overhauling and repair. A preceding table (p. 151) shows the wide difference between "running" expense and total cost of operation for all dredges of this company for 1913.

HOW OPERATING COSTS MAY INDICATE INEFFICIENCY.

A fact that must be kept in mind is that low operating costs per cubic yard may possibly mean low gold-saving efficiency. Although high yardage records and low operating cost per cubic yard are desirable the operator should strive for the best economic results. In ordinary gravel the winch man can easily crowd the tables, and the desire of the dredge master to get yardage records, as was often the case in early California dredging, may result in considerable unwashed gravel going to the tailing piles. On some dredges a careful study showed that it was advisable to reduce the speed of the buckets in order not to crowd the gold-saving tables, and the net result proved the economic success, at these dredges at least, of more thoroughly washing a smaller amount of gravel instead of crowding the screen and tables with all that the dredge could dig. In the Oroville field a large amount of the ground dug in early days was not thoroughly washed under these conditions and there was consequently a considerable loss of the gold content.

The buckets in use in California range in capacity from 3 to 15 cubic feet and some of the dredges being built in 1916 will have 17-foot buckets. Dredge operators, following California practice, prefer the close-connected bucket line to the open-connected type and also prefer digging spuds to headlines, having demonstrated to their entire satisfaction over a long period of experimenting, the superiority of these methods. In practice, the close-connected bucket line dumps at the rate of 18 to 22 buckets a minute, whereas the open-connected averages 12 to 14 buckets a minute. On some of the older boats when the change was made from open to close connected buckets it was claimed that the yardage was nearly doubled and the expense per cubic yard greatly decreased.

COST FIGURES FOR CERTAIN DREDGES.

To illustrate the advantage spuds have over headlines the following example may be given: Two 13-foot Natomas dredges, in ground of apparently the same character within 200 yards of each other, one working on spud, the other on headlines, showed a difference in capacity of 100 yards per hour in favor of the spud method. An instance where headlines proved satisfactory was that of the 18-foot dredge of the Boston & Idaho Co. On account of extremely easy digging no spuds have been used, though the dredge was designed with spud casings should operation have shown the advisability of adding spuds.

As has been stated, the working costs of a dredge may vary greatly for different years and the results of any one year's operation may not be either higher or lower than the general average. In this connection it is interesting to have the figures of operation for over the entire life of a dredge on a property.

Pacific No. 1, a 7½-foot close-connected dredge, went into commission in May, 1906, and worked continuously until the hull sprang a leak and sank December 3, 1911. The hull was repaired and the dredge continued at work until July 14, 1913, when it was dismantled. The machinery was then moved to Mammoth Bar and placed on a new hull, where it is now working. During the seven years of operation at Oroville the dredge handled a total of 9,500,000 cubic yards, from an average depth of 34 feet. Figures of cost for the entire period are furnished as follows:

Average working costs of Pacific No. 1 dredge.

Labor	$0.011
Power	.007
Repairs and supplies	.015
Taxes and insurance	.002
Miscellaneous	.002
Depreciation [a]	.007
Total	.044

Even more interesting than the figures above are the figures for a 3½-foot dredge working under average Oroville conditions for 7½ years.

Average working costs of a 3½-foot dredge.

Dredging time, days	2,248
Cubic yards dredged	3,246,836
Cubic yards per calendar day	1,132
Cubic yards, per operating hour	60
Labor cost, cents per cubic yard	3.0
Power cost, cents per cubic yard	1.8
Water cost, cents per cubic yard	.1
Repairs and supplies, cents per cubic yard	2.3
General expenses, cents per cubic yard	1
Total cost per yard	8.2

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[a] Depreciation was figured at a rate to return the cost of the dredge with an estimated life of 10 years.

The original cost of the boat was $52,014.75, and after operations ended the machinery was sold for $6,000, leaving a net cost for the dredge of $46,014.75, or 1.11 cents a cubic yard, making the complete and total cost of operations exclusive of the original cost of property and for prospecting approximately 9.3 cents a cubic yard of material dug.

Complete figures of similar scope for some of the larger dredges are not yet obtainable, as the dredges have not yet finished work on the ground for which they were built. The following table, showing total costs for the four Conrey dredges, however, are of interest as showing original cost of dredges and all charges of working:

Total costs of dredges of Conrey Placer Mining Co.

Item.	Dredge No.			
	1.	2.	3.	4.
Ledger balance, July 31, 1915	$35,129.37	$34,677.77	$66,600.00	$146,000.26
Depreciation to date	69,164.47	68,837.96	118,964.68	150,205.78
Cost of dredges	104,293.84	103,515.73	185,564.68	296,206.04
Charged to profit and loss:				
Raising No. 1	3,418.85			
Raising No. 2		3,893.98		
Reconstruction No. 2		20,778.59		
Reconstruction No. 3			12,880.48	
Total cost of dredges, including all changes and repairs not charged to operating costs	107,712.69	128,188.30	198,445.16	296,206.04
Total yardage from starting to July 31, 1915	7.817,520	5,515,756	8,166,281	9,880,333
Total operating costs from starting to July 31, 1915 [a]	$442,530.53	$382,739.41	$550,974.35	$479,115.94
Total in cents per yard	5.66	6.94	6.75	4.85
Upkeep of chain from starting to July 31, 1915	$80,500.00	$82,100.00	$123,000.00	$107,000.00
Total in cents per yard	1.03	1.49	1.51	1.08
Total yardage from starting to July 31, 1915, all dredges				31,379,890
Total operating costs from starting to July 31, 1915, all dredges				$1,855,360.23
Total in cents, per yard				5.91
Total cost of upkeep of chain from starting to July 31, 1915				$392,600.00
Total, in cents, per yard				1.25

[a] Including upkeep chain.

NOTE.—If the dredges were written off with above yardage, the total cost of operating would be increased by 2.32 cents a yard, but as the dredges have considerable life remaining, the cost of plant per yard will be considerably less than this figure.

The chief disadvantage of dredging in Montana as compared with California is the severity of the winter, the thermometer falling as low as 30° to 40° F. below zero. The company continues work throughout the year, except at No. 3 dredge, where the long sluice would make heating very expensive. Each dredge has a steam plant which is used to heat the boat and to furnish steam for attacking the ice that accumulates on ladders, rollers, sheaves, etc., and the frozen material that gathers in the hopper; during the coldest weather the suction of the dredge pumps will freeze if the dredge is shut down for short periods. Plate LXII shows the Conrey No. 3 dredge in winter.

The large boat of the Boston & Idaho Dredging Co. began work in May, 1911, and to December 31, 1915, had handled 17,417,894 cubic yards at a cost of 2.772 a cubic yard. The buckets were changed from 14 to 18 cubic feet in the latter part of 1913, greatly increasing the yardage; the amount of ground handled being 3,517,000 cubic yards in 1913, and 4,670,124 cubic yards in 1914.

Working costs for the entire period of operation are segregated as shown below. Power is generated at the company's hydroelectric plant and is charged at $4 a horsepower a month. Maintenance and repairs is an arbitrary figure for covering suspense account and charging off all supplies on hand.

Operating costs, Boston & Idaho Dredging Co., 1914.

	Total.	Per cubic yard.
		Cents.
Labor	$69,731.86	0.400
Power	113,317.15	.651
Maintenance and repairs	211,429.48	1.214
Supplies	12,322.51	.071
Superintendence and office, stable	47,946.08	.275
Sundries	6,184.33	.036
Taxes and insurance [a]	15,276.95	.089
Drilling, total charge	6,206.89	.036
Total cost	482,415.25	2.772

The depth of ground ranges from 14 to 36 feet. Digging is generally easy, but in places large bowlders and buried trees are encountered; the latter are either cut through by the buckets or so loosened as to float to surface. The original cost of the dredge was approximately $225,000, so although the dredge is still at work and the total yardage to be handled on the present property is not yet determined, it will be shown that the cost of the dredge per cubic yard dug will probably not exceed 1 cent.

The cost of electric power varies greatly. Some companies can obtain favorable rates because of the favorable situation of their properties or the feasibility of otherwise arranging their own proper plant. Some companies, such as the Boston & Idaho Co., operate their own power plant. The Boston & Idaho Co. makes a bookkeeping charge of $4 a horsepower-month for the power consumed on its dredge.

For the large companies power rates range from $\frac{3}{4}$ cent to $\frac{4}{5}$ cent a kilowatt-hour; for the smaller companies in different fields the power rate may be 1½ cents or more.

The power consumption and cost of power per cubic yard varies because of difference in first cost of electricity, and, in greater proportion, because of the conditions encountered in digging. The following table of power cost per cubic yard illustrates these variations:

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[a] Taxes do not include United States income tax or State "net profit" tax.

Power cost for dredges, 1914.

Dredge	Bucket capacity.	Average depth dug.	Yardage handled.	Power cost per cubic yard.
	Cu. ft.	*Feet.*	*Cu. yds.*	*Cents.*
Boston & Idaho	18	30	4,670,124	0.58
American Division No. 1, Pacific	7½	....	165,871	.75
Oroville Division No. 4	7½	....	1,353,594	.71
Natomas No. 4	13½	22	2,049,853	.50
Natomas No. 9	15	54	2,346,824	1.41
Feather No. 1	7½	33	1,815,079	.54
Yuba No. 6	7½	48	1,927,804	.66
Yuba No. 14	16	64	3,242,359	1.01
Conrey No. 4	16	50	3,108,933	.98
Conrey No. 3[a]	9½	34	1,068,144	1.44
Marysville No. 4	8	63	2,112,654	.78
Tonopah No. 1	9½	(?) 38	919,397	1.2
Tonopah No. 3[a]	5	(?) 45	743,120	2.4

[a] Flume dredge with a high power consumption.

As an example of the working costs of a small dredge in an isolated district, the following figures obtained from the manager's report are given. The figures should be of especial interest to the small operator or to persons who contemplate financing properties in the far north.

Costs at Berry dredge, Seward Peninsula, Alaska.

Operating time, June 3 to October 17; 3½ cubic feet buckets; operating days, 131; operating, 74.3 per cent; total ground handled, 213,355 cubic yards; average depth, 11.9 feet; some frozen ground; wood used as fuel cost $10 a cord on the bank.

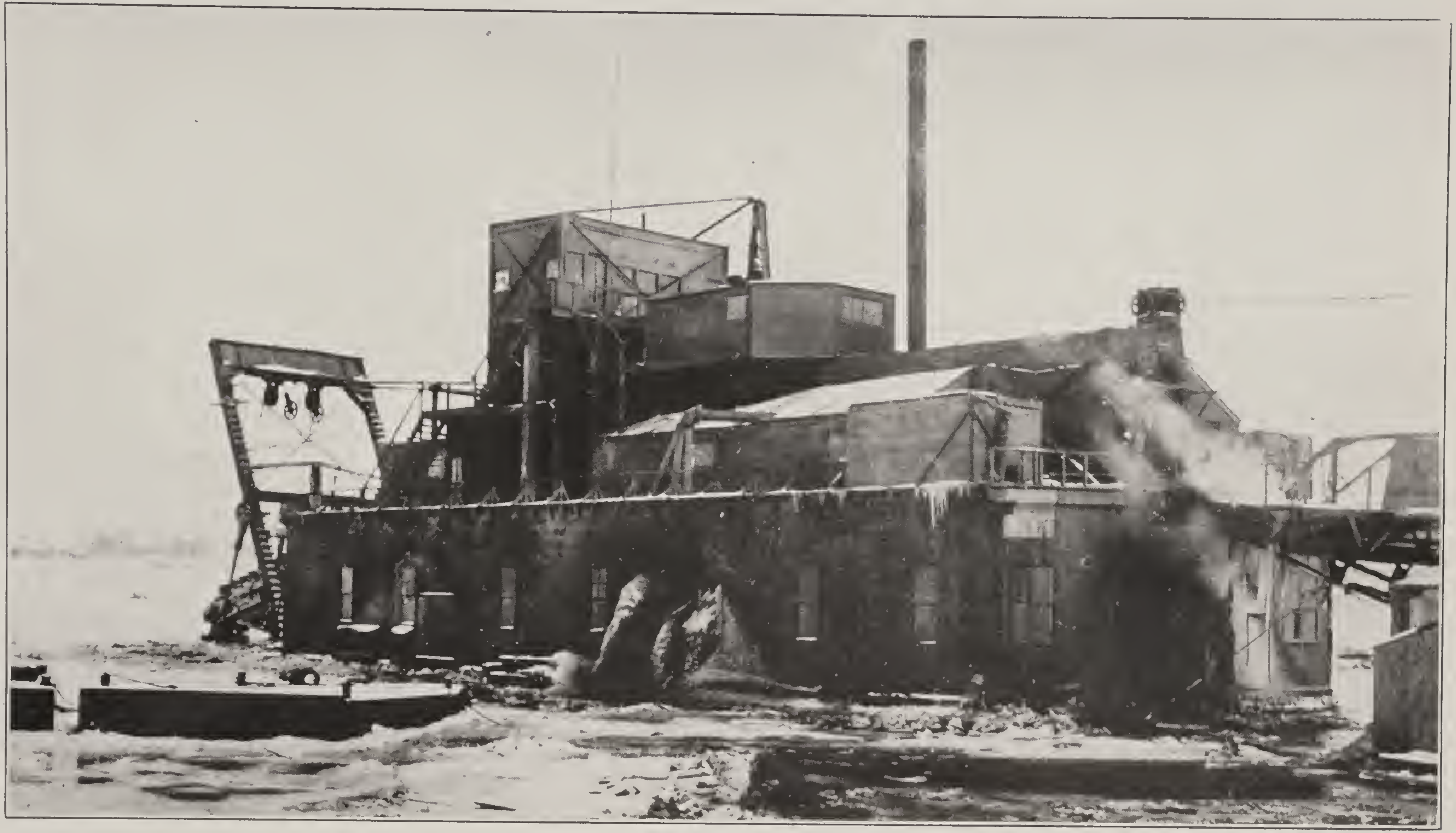
AMORTIZATION CHARGES INCLUDING GENERAL EXPENSE (estimated).

Items.	Total.	Cost per cubic yard.
		Cents.
Dredge labor, including traveling expenses	$12,565.46	5.89
Repairs and supplies used, 1915	3,682.09	1.72
Wood used, 1915	5,499.76	2.57
Organization expense apportioned to 1915	87.69	.04
Insurance and taxes	623.25	.28
Bullion express	822.37	.38
Office expense	721.56	.34
Total	24,002.18	11.24

ACTUAL EXPENSE FOR DREDGE OPERATION.

	Cents.
Depreciation of equipment, taken at $14,000 a year	6.56
Interest on investment ($161,000), $9,660 a year	4.52
Preparation of ground and general expense, divided among 600,000 cubic yards, $24,091.99	4.01
Total	15.09
Estimated gross expense per cubic yard	26.33

Mr. Hamil reports that the cost of boarding the whole force of men was 4,039 man-days at the rate of $2.02 a man-day $8,158.78



CONREY NO. 3 DREDGE IN WINTER.

In writing this report and preparing the tables that show the segregation of working costs it was deemed advisable to use the individual tables of the different companies, because the different methods of apportioning costs render difficult the making of uniform tables for dredges under different managements. In addition, the author presents a table prepared for a paper read before the American Institute of Mining Engineers in 1912.[a] The costs given therein are practically the same as would prevail under similar conditions to-day.

A paper on dredging would be incomplete without inclusion of some reference to the work at the Panama Canal. According to W. G. Combey, in a paper on dredging in the Panama Canal presented at a meeting of the International Engineers' Congress in San Francisco, September, 1915, the grand total of material removed from the Panama Canal and auxiliary works by dredging from 1905 to December, 1914, was 115,700,000 cubic yards. The total cost, which includes work of ladder, dipper, and suction dredges and the total cost of operation, maintenance, material, and plant and all administrative charges, was 26.77 cents a cubic yard.

Operators of gold dredges have been interested in the performance of a large bucket-ladder dredge that had two lines of buckets. In one line the buckets had a capacity of 1 cubic yard, in the other 2 cubic yards. These bucket lines were interchangeable, one being used for heavy digging and the other where the work was lighter. The total material handled by this dredge from April, 1912, to January, 1915, is given at 335,000 cubic yards, and the average cost as 36.59 cents a yard.

The material from bucket, dipper and sea-going suction dredges was delivered to barges and hauled 4 to 10 miles to permanent dump. The intermittent nature of the work and the long haul accounts in a measure for the seemingly high operating cost. The table following prepared from Mr. Combey's report is given for such interest as it may have. Comparisons between the work of these dredges and those operating in different placer fields is of course futile.

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[a] Janin, Charles, Present-day problems in California gold dredging: Trans. Am. Inst. Min. Eng., vol., 42, 1912, p. 873.

Operating costs of Panama dredges.

Kind of dredge.	Capacity of buckets.	Depth dug.	Digging time.	Cubic yards handled.	Cost per cubic yard.[a]	Remarks.
	Cu. yds.	*Feet.*	*Months.*		*Cents.*	
Sea-going ladder	2	45	34	3,351,633	36.61	Material: Hard clay and rock; haul 3 to 4 miles.
Dipper	5	5	12	743,839	29.4	Mud, stiff clay, and rock; haul 3 to 4 miles.
Do	5	45	12	420,152	34.3	Mud, clay, coral rock, and rock; haul 5 to 6 miles.
Do	15	35.47	10	1,228,717	[b] 42.4	Hard and soft rock; haul 9 to 10 miles.
Do	15	35.47	8	939,508	[c] 42.9	Do.
	In. suc.					
Pipe-line suction	20	42	10	562,015	23.9	
Do	20	42	12	1,204,520	21.2	Mud, clay, gravel, and broken rocks, 3,000 feet; clay and broken rock; pipe line 7,000 feet long, using one relay.
Suction dredge delivering to barges.	....	45	12	3,099,022	10.9	Mud, sand, and clay; haul 4 to 5 miles.
Total quantity all material removed by dredging, 1905 to January, 1915.	....	....	....	115,722,733	26.7	

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[a] Costs are for material placed on dump and include all operations, maintenance, material, plant, and administrative charges. Cost of plant on arbitrary figure assessed per cubic yard. Cost of different dredges approximated is as follows: 2-yard ladder, $400,000; 5-yard dipper, $95,000; 20-inch sea-going suction, $330,000; 20-inch pipe suction, $100,000 to $160,000.

[b] A credit of $23,680 for unused material has been allowed.

[c] A credit of $10,948 for unused material has been allowed.

Data on operating costs of dredges of the Yuba Consolidated Goldfields are shown in the table following.

Operating reports of Yuba Consolidated Goldfields for year ended February 28, 1915.

Dredge No.	Total dredging time.		Average daily time.		Per cent of possible running time.	Ground dredged.	Daily average.	Average depth.	Operation.										Insurance.		Taxes.		General expense.		Total expense.	
									Labor, cost per cubic yard.	Labor, per cent of total cost.	Material, cost per cubic yard.	Material, per cent of total cost.	Power, cost per cubic yard.	Power, per cent of total cost.	Repair labor, cost per cubic yard.	Repair labor, per cent of total cost.	Repair material, cost per cubic yard.	Repair material, per cent of total cost.	Cost per cubic yard.	Per cent of total cost.	Cost per cubic yard.	Per cent of total cost.	Cost per cubic yard.	Per cent of total cost.	Cost per cubic yard.	Per cent of total cost.
	Hrs.	*min.*	*H.*	*m.*		*Cu. yds.*	*Cu. yds.*	*Feet.*	*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*		*Cts.*	
3 [a]	3,095		20	22	85.1	434,765	2,860	64.3	1.54	30.1	0.07	1.3	1.04	20.4	0.40	7.8	1.26	24.7	0.14	2.7	0.13	2.6	0.53	10.4	5.11	.....
3 [a]	2,407	20	20	45	86.8	395,265	3,407	67.8	1.29	27.9	.03	0.6	1.21	26.2	.27	5.9	.91	19.6	.00	0.0	.21	4.5	.71	15.3	4.63	.....
4 [b]	1,211	15	20	53	87.1	205,908	3,550	63.1	1.19	17.4	.12	1.8	1.04	15.1	.91	13.2	2.83	41.2	.20	3.0	.11	1.6	.46	6.7	6.86	.....
4 [b]	3,453		21	11	88.5	632,667	3,881	65.1	1.13	31.8	.03	.09	1.00	28.3	.19	5.4	.39	10.9	.00	0.0	.18	5.2	.62	17.5	3.54	.....
6	7,504	55	20	40	86.3	1,927,805	5,310	48.1	.84	25.5	.03	.09	.66	20.1	.38	11.4	.96	29.3	.06	1.8	.07	2.2	.29	8.8	3.29	.....
7	7,742	25	21	20	89.0	1,485,390	4,091	67.4	1.08	19.2	.05	.09	.88	15.7	.53	9.5	2.52	44.9	.07	1.3	.10	1.7	.38	6.8	5.61	.....
8	7,859	20	21	39	90.4	1,705,750	4,699	77.6	.93	18.4	.05	1.0	.83	16.5	.53	10.4	2.23	44.2	.07	1.3	.08	1.6	.33	6.6	5.05	.....
9	7,426	50	20	28	85.4	1,215,740	3,349	67.0	1.32	22.9	.06	1.0	1.16	20.0	.64	11.0	1.93	33.3	.11	1.9	.11	1.9	.46	8.0	5.79	.....
11	7,276	50	20	3	83.7	1,315,306	3,623	63.0	1.17	24.2	.05	1.1	1.06	21.8	.65	13.5	1.27	26.2	.10	2.1	.11	2.2	.43	8.9	4.84	.....
12	7,213	5	19	52	83.0	1,541,379	4,246	52.1	.99	18.7	.04	.08	.91	17.2	.78	14.7	2.03	38.3	.09	1.7	.09	1.8	.36	6.8	5.29	.....
13	7,125	5	19	37	82.0	2,946,018	8,116	55.4	.66	17.0	.09	2.3	1.01	26.0	.36	9.2	1.30	33.6	.14	3.5	.13	3.3	.20	5.1	3.89	.....
14	7,740	15	21	19	89.1	3,242,359	8,932	70.0	.64	18.7	.08	2.4	1.04	30.2	.35	10.1	.92	26.8	.00	0.0	.23	6.7	.17	5.1	3.43	.....
Total	70,055	20	......		.....	17,048,352																				
Average	....		20	39	86.2	....	5,024	61.4	.92	20.8	.06	1.4	.95	21.7	.47	10.7	1.47	33.4	.08	1.7	.13	3.0	.32	7.3	4.40	.....

[a] Dredge No. 3 dismantled July 31, 1914; reconstructed in steel, started work Nov. 5, 1914.
[b] Dredge No. 4 dismantled Apr. 27, 1914; reconstructed in steel, started work Sept. 18, 1914.

LABOR COSTS.

The greater efficiency and larger capacity of the modern, as compared with the earlier, dredges have steadily decreased labor costs per cubic yard in spite of shorter working hours and higher wages for the men. On some of the early dredges the labor alone cost as much per cubic yard as the total operating charges at present.

Wage scales vary somewhat in different localities, but are approximately the same in different areas in the same State. The following table gives the approximate scale of wages in California, Montana, and Alaska. In the latter country board is generally included, or an additional allowance is made for it.

Approximate scale of wages at dredges.[a]

	California.	Montana.	Alaska.
Dredge masters	$150 a month	$6 a day	$200 a month.
Winchmen	$4 a day	$5 a day	50 to 60 cents an hour.
Oilers	$3 to $3.50 a day	$3.50 a day	
Shoremen	$2.50 a day	$3.50 a day	
Helpers	$2.50 a day	$2.50 a day	

[a] Since this was compiled wages have advanced in some localities.

An indication of the number of men employed in dredging is the total force required by the Conrey Co. in different departments for the normal working of its dredges and for company business. The force is distributed as follows:

Men employed in dredging by Conrey Co.

Dredge crew No. 1 dredge	8
Dredge crew No. 2 dredge	8
Dredge crew No. 3 dredge	10
Dredge crew No. 4 dredge	10
Office and dwellings	4
Transformer station and pole line	2
Machine shop	8
Clean-up crew	3
Laborers	22
Miscellaneous	10
Total	85

In California the Natomas Consolidated employs a total of about 1,500 men in all branches of its operations.

ACCOUNTING METHODS OF NATOMAS CONSOLIDATED.

To illustrate the bookkeeping methods employed, the following list of account numbers used by the Natomas Consolidated and the distribution of repair charges for two of the dredges for the month of August, 1914, are presented·

JANUARY 1, 1915.

Gold-dredging department.

NATOMA DIVISION.

LIST OF ACCOUNT NUMBERS.

OPERATING "NATOMA NO. 1" DREDGE:

Running expense—	
11	Labor.
12	Material.
13	Power.
14	Water.
Repairs—	
15	Labor.
16	Material.
General expense—	
17a	Restraining débris.
b	Levee.
c	Fire protection.
d	Accident insurance.
e	Division expense (including pro rata of company, Sacramento, auditing and purchasing expenses.
General items—	
18a	Taxes.
b	Insurance.
c	Depreciation.

Natomas Consolidated of California.

DREDGE OPERATING COSTS.

Distribution of Labor and Material for Repairs, August, 1914.

NATOMA DIVISION.

Dredge No. 1:			
Labor			$574.00
Supplies—			
Miscellaneous lumber	$28.90		
3 return idlers	30.72		
2 wrecking sheaves and shackles	51.12		
1 G. E. 50 h. p. motor No. 88209	60.00		
12 forged U. T. heel plates	48.00		
13 perforated shaker plates	278.82		
4 motor bearings	41.50		
1 10-inch C. T. pump head, suction side	49.40		
750 feet 1-inch Leschen wire rope	236.63		
Sundries	144.73		
Hauling	93.60		
		$1,063.42	
Natoma shop—			
Repairs to main hopper	33.80		
Repairs, general	21.62		
		55.52	
			1,118.84
Total			1,692.84
Dredge No. 2:			
Labor			$239.09
Supplies—			
100 bucket bushings	247.86		
24 bucket pins	524.16		
2 ladder rollers	112.04		
750 feet of ⅞-inch 6/19 Roebling wire rope	94.58		
90, 30, 40¼, 32, and 43½ riffles	100.30		
24 rib bars, job 569	95.76		
Sundries	136.27		
Hauling	145.71		
		1,456.68	
Natoma shop, repairs, general		11.21	
			1,467.89
Total			1,706.98

MISCELLANEOUS COST TABLES AND FORMS.

The tables and forms following need no special comment, as they are self-explanatory. The figures presented should prove of considerable value. Moreover the forms used for recording them are of interest.

Operating costs of dredges of

OROVILLE DIVISION, DREDGE NO. 1.

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Total costs.					
					Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.
1	Total or average, 7 months, 1906.	3,773	602,803	160	$7,742.75	$6,227.64	$7,566.91	$1,489.67	$1,220.10	$5,250
2	Total or average, 1907.	6,889	1,109,226	161	13,408.15	11,148.10	18,245.08	2,553.72	4,974.31	9,000
3	Total or average, 1908.	6,774	1,365,892	202	14,040.98	8,980.70	21,949.06	3,712.79	2,691.99	9,000
4	Total or average, 1909.	7,048	1,215,214	172	14,590.22	8,548.69	18,857.95	2,113.63	2,878.94	9,000
5	Total or average, 1910.	7,054	1,488,503	211	14,403.01	9,124.97	23,691.37	1,460.40	2,928.68	9,000
6	Total or average, 1911.	6,616	1,495,331	226	15,571.36	8,419.87	19,227.34	2,920.80	2,938.43	9,000
7	Total or average, 1912.	6,687	1,498,504	224	14,914.08	9,649.80	27,690.23	2,920.80	4,905.26	9,000
8	Total or average, 6 months, 1913.	3,722	730,746	196	6,636.40	5,092.20	3,202.18	1,460.40	1,896.90	4,500
OROVILLE DIVISION, DREDGE NO. 2.										
9	Total or average, 7 months, 1906.[1]	3,699	217,274	59	$6,543.19	$3,406.12	$7,186.94	$350.00	$1,477.95	$2,331
10	Total or average, 1907.[2]	6,148	469,253	76	10,226.90	4,827.27	13,636.32	600.00	4,942.29	3,996
11	Total or average, 1908.	6,808	709,508	104	10,839.62	4,511.10	15,834.90	1,550.07	2,626.46	3,996
12	Total or average, 1909.	5,487	609,442	111	11,731.75	3,859.80	5,434.14	1,162.72	2,931.62	3,996
13	Total or average, 1910.	7,116	683,198	96	11,237.88	5,417.45	7,890.57	835.20	2,964.44	3,996
14	Total or average, 1911.	6,792	757,091	111	10,791.12	4,626.80	13,961.53	1,531.20	2,868.05	3,663
OROVILLE DIVISION, DREDGE NO. 3.										
15	Total or average, 7 months, 1906.	4,265	506,284	181	$6,594.51	$4,902.48	$6,138.54	$551.25	$1,374.72	$3,500
16	Total or average, 1907.	7,002	884,203	126	11,363.63	7,627.20	13,965.27	945.00	5,358.78	6,000
17	Total or average, 1908.	7,453	989,966	133	11,013.23	6,914.80	15,242.28	2,578.72	2,722.08	6,000
18	Total or average, 1909.	6,964	803,429	115	12,032.39	5,709.75	13,481.67	1,510.02	2,884.49	6,000
19	Total or average, 1910.	7,162	916,194	128	11,992.46	6,434.45	12,309.03	1,044.00	2,908.30	6,000
20	Total or average, 1911.	7,265	712,142	98	12,135.82	7,333.80	10,879.58	2,088.00	2,948.33	6,000
21	Total or average, 1912.	5,727	688,733	120	8,853.23	5,865.61	8,354.80	1,566.00	2,576.91	4,500

[1] Operating on tract M, 30 per cent royalty.
[2] Operating 4 months on tract M, 30 per cent royalty.

Pacific Gold Dredging Co.

Total costs—Contd.		Costs per yard, cents.								Acres worked.	Item No.
Express and treatment.	Total.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.	Express and treatment.	Total.		
OROVILLE DIVISION, DREDGE NO. 1.											
$206.62	$29,703.69	1.284	1.033	1.255	0.347	0.202	0.870	0.034	4.927	11.26	1
319.41	59,648.67	1.209	1.005	1.645	.230	.449	.811	.029	5.378	19.95	2
438.06	60,813.58	1.028	.657	1.607	.272	.197	.659	.032	4.452	23.47	3
352.31	56,341.74	1.201	.703	1.552	.174	.237	.740	.029	4.636	21.82	4
376.68	60,985.11	.967	.613	1.592	.098	.197	.605	.025	4.097	28.09	5
314.57	58,392.37	1.041	.563	1.286	.195	.197	.602	.021	3.905	32.66	6
280.48	69,360.65	.995	.644	1.848	.195	.328	.600	.019	4.629	27.10	7
135.18	22,923.26	.908	.697	.438	.200	.259	.616	.019	3.137	11.61	8
OROVILLE DIVISION, DREDGE NO. 2.											
$101.22	$21,396.42	3.011	1.567	3.308	0.161	0.680	1.073	0.046	9.846	....	9
184.96	38,413.74	2.179	1.029	2.906	.128	1.053	.852	.039	8.186	8.32	10
305.68	39,633.83	1.528	.636	2.232	.218	.370	.563	.043	5.590	15.16	11
149.38	29,265.41	1.925	.633	.892	.191	.481	.656	.024	4.802	13.76	12
182.98	32,524.52	1.645	.793	1.155	.122	.434	.585	.027	4.761	14.26	13
164.78	37,606.48	1.425	.611	1.844	.202	.379	.484	.022	4.967	13.83	14
OROVILLE DIVISION, DREDGE NO. 3.											
$166.90	$23,228.40	1.303	0.968	1.213	0.109	0.272	0.691	0.033	4.589	9.81	15
233.12	45,493.00	1.285	.863	1.579	.107	.606	.679	.026	5.145	18.52	16
269.44	44,740.55	1.112	.698	1.540	.260	.275	.606	.027	4.518	18.13	17
235.19	41,853.51	1.498	.711	1.678	.188	.358	.747	.029	5.209	15.36	18
285.47	40,973.71	1.309	.702	1.343	.114	.318	.655	.031	4.472	16.73	19
185.94	41,571.47	1.704	1.030	1.528	.293	.414	.843	.026	5.838	12.36	20
141.31	31,857.87	1.285	.852	1.213	.227	.374	.653	.021	4.625	11.73	21

Operating costs of dredges of Pacific

Item No.	Period covered.	Hours run.	Cubic yards worked.	Yards per hour.	Total costs.					
					Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.
	OROVILLE DIVISION, DREDGE NO. 4.									
22	Total or average, 7 months, 1906.	......	....	.....	....	....	....	....	....	....
23	Total or average, 1907.	......	....	.....	....	....	....	....	....	....
24	Total or average, 1908.	6,321	1,491,910	236	$13,040.95	$9,704.07	$14,447.31	$3,499.88	$3,582.60	$9,163
25	Total or average, 1909.	7,091	1,371,263	193	14,410.53	9,120.75	18,918.69	2,113.63	2,884.07	9,996
26	Total or average, 1910.	7,410	1,383,332	187	14,140.09	9,723.38	21,350.21	1,460.40	2,928.81	9,996
27	Total or average, 1911.	7,372	1,383,795	188	14,627.24	10,225.88	12,613.20	2,920.80	2,920.38	9,996
28	Total or average, 1912.	6,999	1,556,579	222	14,051.40	10,098.04	19,900.62	2,920.80	4,125.40	9,996
29	Total or average, 1913.	7,501	1,444,533	192	13,519.77	10,866.20	19,501.08	4,140.88	5,487.95	9,996
	1914.									
30	January	611	99,104	162	1,480.55	870.67	1,669.28	307.45	407.91	833
31	February	518	104,866	202	1,100.00	682.40	1,229.00	326.16	360.58	833
32	March	666	128,574	193	1,304.55	882.20	1,952.65	326.16	351.79	833
33	April	647	116,883	181	1,118.55	864.00	1,327.99	326.16	366.19	833
34	May	549	120,218	219	1,249.07	720.00	883.51	334.12	352.95	833
35	June	461	95,990	208	1,433.32	587.20	1,436.91	332.25	374.98	833
36	July	635	113,952	180	1,111.09	802.20	222.83	254.63	386.43	833
37	August	652	111,176	170	966.30	858.00	1,148.94	192.13	275.01	833
38	September	635	117,755	185	1,219.20	834.40	174.24	323.38	275.37	833
39	October	655	125,902	192	1,111.58	894.40	840.00	257.74	286.84	833
40	November	500	99,582	199	1,277.40	682.80	1,362.40	257.74	421.37	833
41	December	660	119,592	181	1,157.35	898.20	1,815.49	200.84	287.70	833
42	Total or average	7,189	1,353,594	188	14,528.96	9,576.47	14,063.24	3,438.76	4,147.12	9,996
	AMERICAN DIVISION, DREDGE NO. 1.									
43	Total or average, 3 months, 1913.	1,801	374,992	208	$4,282.24	$2,892.00	$4,530.53	$180.81	$2,047.21	$3,000
	1914.									
44	January	518	119,368	230	1,364.07	884.00	1,444.09	157.49	663.91	1,000
45	February	463	87,783	190	1,082.59	924.00	2,256.55	129.17	639.74	1,000
46	March	652	122,970	190	1,206.93	944.00	3,910.88	129.17	627.39	1,000
47	April	667	150,521	226	1,169.04	1,204.00	1,056.86	129.17	652.39	1,000
48	May	668	156,663	234	1,196.72	1,104.00	1,009.42	133.32	594.22	1,000
49	June	661	171,510	259	1,178.70	1,092.00	1,916.28	133.32	617.67	1,000
50	July	678	133,614	197	1,336.95	1,068.00	3,713.07	218.74	605.17	1,000
51	August	660	123,457	187	1,179.83	1,194.80	1,481.91	156.23	496.47	1,000
52	September	600	121,316	202	1,098.55	957.40	365.51	289.57	527.47	1,000
53	October	620	152,103	245	1,202.03	1,165.60	3,254.09	221.84	526.73	1,000
54	November	633	153,400	242	1,136.28	1,046.40	1,449.12	221.84	736.84	1,000
55	December	644	166,012	258	1,210.05	952.00	473.05	151.05	520.65	1,000
56	Total or average	7,464	1,658,717	222	14,361.74	12,536.20	22,330.83	2,070.91	7,208.65	1,200
	BUTTE CREEK DIVISION, DREDGE NO. 3.									
	1914.									
57	January	......	....	.....	....	....	....	....	....	....
58	February	......	....	.....	....	....	....	....	....	....
59	March	......	....	.....	....	....	....	....	....	....
60	April	......	....	.....	....	....	....	....	....	....
61	May	......	....	.....	....	....	....	....	....	....
62	June	......	....	.....	....	....	....	....	....	....
63	July	......	....	.....	....	....	....	....	....	....
64	August	......	....	.....	....	....	....	....	....	....
65	September	......	....	.....	....	....	....	....	....	....
66	October	525	45,156	86	$923.70	$423.36	$218.20	$110.00	$350.89	$500
67	November	545	69,114	127	935.85	580.23	843.02	110.00	468.01	700
68	December	570	71,611	125	1,060.61	538.65	327.73	110.00	399.68	600
69	Total or average	1,640	185,881	113	2,920.16	1,542.24	1,388.95	330.00	1,218.58	1,800

Gold Dredging Co.—Continued.

OROVILLE DIVISION, DREDGE NO. 4.

Total costs—Contd.		Costs per yard, cents.									
Express and treatment.	Total.	Labor.	Power.	Repairs and supplies.	Taxes and insurance.	Miscellaneous expenses.	Depreciation.	Express and treatment.	Total.	Acres worked.	Item No.
....	....	....	....	....	....	....	....	....	....	....	22
....	....	....	....	....	....	....	....	....	....	....	23
$332.63	$53,770.54	0.874	0.650	0.968	0.235	0.240	0.614	0.022	3.603	31.86	24
297.20	57,740.87	1.051	.665	1.379	.154	.211	.729	.022	4.211	27.67	25
361.52	59,960.41	1.022	.703	1.543	.105	.212	.723	.026	4.334	25.00	26
322.63	53,626.13	1.057	.739	.912	.211	.211	.722	.023	3.875	27.06	27
300.17	61,392.43	.903	.649	1.278	.188	.265	.642	.019	3.944	30.73	28
347.71	63,859.59	.936	.752	1.350	.287	.380	.692	.024	4.421	23.20	29
25.59	5,594.45	1.494	.879	1.684	.310	.412	.840	.026	5.645	1.45	30
25.81	4,556.95	1.049	.651	1.172	.311	.344	.794	.024	4.345	1.75	31
60.62	5,710.97	1.015	.686	1.518	.253	.274	.648	.049	4.442	2.19	32
35.67	4,871.56	.957	.739	1.136	.279	.313	.713	.031	4.168	1.61	33
16.49	4,389.14	1.039	.599	.735	.278	.293	.693	.014	3.651	1.56	34
31.00	5,028.66	1.493	.612	1.497	.346	.391	.868	.032	5.239	1.20	35
24.57	3,634.75	.975	.704	.195	.223	.340	.731	.022	3.190	1.66	36
33.42	4,306.80	.869	.772	1.034	.173	.247	.749	.030	3.874	1.73	37
19.58	3,679.17	1.035	.709	.148	.275	.234	.707	.016	3.124	1.67	38
22.51	4,246.07	.883	.710	.667	.205	.228	.662	.018	3.373	1.70	39
17.39	4,852.10	1.283	.686	1.368	.258	.423	.837	.017	4.872	1.19	40
69.07	5,261.65	.968	.751	1.518	.168	.240	.696	.058	4.399	1.74	41
381.72	56,132.27	1.073	.708	1.039	.254	.306	.738	.029	4.147	19.45	42
AMERICAN DIVISION, DREDGE NO. 1.											
$133.51	$17,066.30	1.142	0.771	1.208	0.048	0.546	0.800	0.036	4.551	7.66	43
48.33	5,561.94	1.143	.741	1.209	.132	.556	.838	.040	4.659	2.52	44
19.03	6,051.08	1.233	1.053	2.571	.147	.729	1.139	.021	6.893	1.71	45
20.53	7,838.90	.982	.768	3.180	.105	.510	.813	.017	6.375	2.20	46
26.96	5,238.37	.777	.800	.702	.086	.433	.664	.018	3.480	2.59	47
20.76	5,058.44	.764	.705	.644	.085	.379	.639	.013	3.229	3.13	48
39.84	5,977.81	.687	.637	1.118	.077	.360	.583	.023	3.485	3.12	49
70.32	8,012.25	1.001	.799	2.779	.163	.453	.749	.053	5.997	2.57	50
58.57	5,567.81	.956	.968	1.200	.126	.402	.810	.047	4.509	2.43	51
63.06	4,301.56	.906	.789	.301	.239	.435	.824	.052	3.546	2.57	52
36.21	7,406.50	.790	.767	2.139	.146	.346	.657	.024	4.869	3.64	53
102.89	5,693.37	.741	.682	.945	.144	.480	.652	.067	3.711	3.45	54
165.06	4,471.86	.729	.573	.285	.091	.314	.602	.099	2.693	2.67	55
671.56	71,179.89	.866	.756	1.346	.125	.434	.723	.041	4.291	32.60	56
BUTTE CREEK DVISION, DREDGE NO. 3.											
....	....	....	....	....	....	....	....	....	....	....	57
....	....	....	....	....	....	....	....	....	....	....	58
....	....	....	....	....	....	....	....	....	....	....	59
....	....	....	....	....	....	....	....	....	....	....	60
....	....	....	....	....	....	....	....	....	....	....	61
....	....	....	....	....	....	....	....	....	....	....	62
....	....	....	....	....	....	....	....	....	....	....	63
....	....	....	....	....	....	....	....	....	....	....	64
....	....	....	....	....	....	....	....	....	....	....	65
$9.82	$2,535.97	2.045	0.938	0.483	0.244	0.777	1.107	0.022	5.616	1.85	66
17.39	3,654.50	1.354	.839	1.220	.119	.677	1.013	.025	5.287	2.16	67
36.86	3,073.53	1.481	.752	.458	.184	.888	.838	.051	4.292	2.38	68
64.07	9,264.00	1.571	.830	.747	.188	.855	.969	.034	4.983	6.39	69

Balance sheet used by Conrey Placer Mining Co.[a]

Assets.			Increase.	Decrease.
Cash mines bank				
Cash mines office				
Cash with treasurer				
Bullion in hand				
Accounts receivable (list)				
Stock—warehouse (list)				
Stock—fuel				
Prepaid accounts				
Dredge No. 1				
Dredge No. 2				
Dredge No. 3				
Dredge No. 4				
Machine shop and tools				
Minor personal property				
Lands, buildings, etc				
Substation and pole line				
Deferred charges (list)				

Liabilities.			Increase.	Decrease.
Accounts payable (list)				
Notes payable				
Accounts accruing				
Capital stock				
Reserve for upkeep of chain				
Surplus account				

.... *Bookkeeper.* .... *Manager.*

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[a] Jennings, Hennen, History and development of gold dredging in Montana: Bull. 121, Bureau of Mines, 1916, p. 25.

Profit-and-loss report used by Conrey Placer Mining Co.[a]

Profit-and-Loss Report for Month of ....

Cubic yards dredged.	Month of............	Year to date...months.
Dredge No. 1		
Dredge No. 2		
Dredge No. 3		
Dredge No. 4		
Total		

Operating costs.		Average per yard.	Months.	Average per yard.
Dredge No. 1				
Dredge No. 2				
Dredge No. 3				
Dredge No. 4				
Total				
Indirect expenses.				
Salaries, mines				
Salaries, Boston				
Expenses, Boston				
Interest				
Insurance				
Taxes				
Telephones				
Office expense				
Building repairs				
Living quarters				
Miscellaneous expenses				
Stable expense				
Maintenance substation and pole line				
Roads, ditches, and fences				
Prospect drilling				
Ranch expense				
Total indirect expenses				
Total operating costs				
Total expenses				
Balance net earnings to surplus account				
Net bullion and earnings.				
Dredge No. 1				
Dredge No. 2				
Dredge No. 3				
Dredge No. 4				
Net bullion, total				
Ranch returns				
Rents				
Lights				
Scrap				
Interest receipts				
Shop receipts				
Miscellaneous receipts				

.... *Bookkeeper.* .... *Manager.*

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[a] Jennings, Hennen, work quoted, p. 26.

Form of operating report used by Conrey Placer Mining Co.[a]

Operating Report for Month of .... Dredge No. ....

Time.	Month of	Year to date	months.
Running hours........................	per cent.	Hours..............................	per cent.
Lost hours...........................	per cent.	Hours..............................	per cent.
Total hours..........................	100 per cent.	Hours..............................	100 per cent.

Excavation.		Months.
Cubic yards dredged..............................virgin..		
Cubic yards dredged..............................tailings..		
Cubic yards dredged..............................gross..		
Average per running hour..............................		
Average per bucket..............................		
Buckets per minute..............................		
Average kilowatt-hours per yard..............................		
Acres dredged..............................		
Average depth to bedrock..............................		

Production.		Average per yard.	Months.	Average per yard.
Gross bullion..............................				
Less express..............................				
Net bullion to profit-and-loss report..............				
Operating cost.				
Labor..............................				
Fuel..............................				
Power..............................				
Supplies..............................				
Upkeep of chain..............................				
Repairs:				
Digging appliances, labor....................				
Digging appliances, material....................				
Pumping appliances, labor....................				
Pumping appliances, material....................				
Gold saving and washing appliances, labor.........				
Gold saving and washing appliances, material......				
Electrical appliances, labor....................				
Electrical appliances, material....................				
Heating appliances, labor....................				
Heating appliances, material....................				
Hull and housing, labor....................				
Hull and housing, material....................				
Miscellaneous....................				
Totals to profit-and-loss report....................				

.... *Bookkeeper.* .... *Manager.*

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[a] Jennings, Hennen, work quoted, p. 27.

Summary of yearly yardage and expenses, Conrey Placer Mining Co., Sept. 1, 1906, to July 31, 1915.[a]

Year.	Cubic yards excavated.	Average operating costs per cubic yard, in cents.							Indirect expenses.[b]	Total.
		Labor.	Fuel.	Power.	Supplies.	Upkeep of chain.	Repairs.	Total.		
1906	322,174	1.65	0.08	1.59	0.20	1.40	1.07	5.99	3.80	9.79
1907	1,152,478	1.34	.06	1.34	.04	.87	1.12	4.77	2.63	7.40
1908	2,248,576	1.45	.08	1.69	.09	1.50	1.45	6.26	1.12	7.38
1909	2,429,524	1.60	.11	1.30	.09	1.69	1.89	7.61	1.22	8.83
1910	2,670,354	1.66	.13	1.29	.10	1.53	1.97	6.88	1.11	7.99
1911	3,566,691	1.29	.12	1.22	.09	1.31	1.80	6.11	1.04	7.15
1912	4,463,354	1.41	.17	1.34	.13	1.37	2.34	6.82	1.05	7.87
1913	4,625,436	1.37	.12	1.36	.09	1.09	1.93	5.96	.92	6.88
1914	6,268,626	1.13	.07	1.15	.09	.94	1.25	4.63	.73	5.36
1915 [c]	3,632,677	1.13	.06	1.15	.08	.98	1.65	5.05	.94	5.99
Total	31,379,890	1.37	.11	1.32	.10	1.00	1.76	5.91	1.05	6.96

[a] Jennings, Hennen, work quoted, p. 28.

[b] Includes salaries, Ruby and Boston, insurance, taxes, maintenance of pole line and substation, roads, ditches, bonus to employees, legal expenses, accidents, all office expenses, etc., but interest on loans and other indebtedness is excluded.

[c] To July 31.

Operating report of Conrey Placer Mining Co., covering electric dredges Nos. 1, 2, 3, and 4, from starting to July 31, 1915.[a]

TIME.

Item.	Dredge No. 1.	Dredge No. 2.	Dredge No. 3.	Dredge No. 4.	Total.
Running hours, per cent	71	67	65	65	67
Lost hours, per cent	29	33	35	35	33
Total	100	100	100	100	100
EXCAVATION DATA.					
Cubic yards dredged, virgin ground	7,817,520	5,515,756	8,166,281	9,880,333	31,379,890
Cubic yards dredged, tailings [b]	72,678	752,240	479,872	145,192	1,449,982
Cubic yards dredged, gross	7,890,198	6,267,996	8,646,153	10,025,525	32,829,872
Average per running hour	174	132	161	420	195
Average per bucket	.18	.18	.19	.38	....
Buckets per minute	15.75	12.98	14.15	18.62	....
Average kilowatt-hours per yard	1.55	1.89	1.73	1.36	....
Acres dredged	223.16	143.62	144.43	134.92	646.13
Average depth to bedrock, feet	22.1	28.1	40.8	45.3	....
OPERATING COSTS, AVERAGE COST PER YARD, IN CENTS.					
Labor	1.50	1.68	1.75	0.79	1.37
Fuel	.11	.13	.13	.07	.11
Power	1.27	1.55	1.48	1.09	1.32
Supplies	.10	.11	.09	.09	.10
Upkeep of chain	1.03	1.49	1.51	1.08	1.25
COST OF REPAIRS, IN CENTS.					
Digging appliances, labor	0.19	0.32	0.28	0.23	0.25
Digging appliances, material	.32	.46	.62	.61	.51
Pumping appliances, labor	.04	.02	.03	.01	.02
Pumping appliances, material	.07	.03	.04	.01	.04
Gold saving and washing appliances, labor	.28	.28	.22	.17	.23
Gold saving and washing appliances, material	.60	.66	.36	.57	.54
Electrical appliances, labor	.04	.05	.05	05	.05
Electrical appliances, material	.03	.04	.06	.04	.04
Heating appliances, labor	.01	....	.01	....	....
Heating appliances, material	.02	.02	.02	....	.01
Hull and housing, labor	.01	.05	.04	.01	.03
Hull and housing, material	.01	.02	.01	.01	.01
Miscellaneous	.03	.03	.05	.02	.03
Total operating costs	5.66	6.94	6.75	4.85	5.91
Indirect expenses, excluding interest [c]	....	....	....	....	1.05
Grand total	....	....	....	....	6.96

[a] Jennings, Hennen, work quoted, p. 29.

[b] The tailings dredged are not used in calculating costs.

[c] In indirect expenses are included salaries, Ruby and Boston, insurance, taxes, maintenance of pole line and substation, roads, ditches, bonus to employees, legal expenses, accidents, all office expenses, etc., but interest on loans and other indebtedness is excluded.

Operating costs of dredges of Natoma

Dredge No.	Size of buckets.	Time.								
		Total days.	Total hours.	Lost.		Dredging.		Efficiency.	Average dredging per day.	
				Hrs.	*min.*	*Hrs.*	*min.*	*P. ct.*	*Hrs.*	*min.*
Natoma Division.										
1	13½	363	8,712	1,606	55	7,105	5	81.6	19	34
2	8½	363	8,712	1,109	25	7,602	35	87.3	20	57
3	8½	364	8,736	1,564	10	7,171	50	82.1	19	42
4	13½	303	7,272	1,204	40	6,067	20	83.4	20	1
5	9	363	8,712	1,542	10	7,169	50	82.3	19	45
6	9	363	8,712	1,452		7,260		83.3	20	
7	9	363	8,712	2,322	30	6,389	30	73.3	17	36
8	15	363	8,712	2,341	30	6,370	30	73.1	17	33
9	15	363	8,712	1,481	55	7,230	5	83.0	19	55
10	15	363	8,712	2,183	55	6,528	5	74.9	17	59
F. R. Division.										
1	7½	363	8,712	902	25	7,809	35	89.6	21	31
2	7½	363	8,712	1,212	35	7,499	25	86.1	20	40
3	13½	363	8,712	1,251	45	7,460	15	85.6	20	33
Natoma Division		3,571	85,704	16,809	10	68,894	50	80.4	19	18
F. R. Division		1,089	26,136	3,366	45	22,769	15	87.1	20	55
Both divisions		4,660	111,840	20,175	55	91,664	5	82.0	19	40

Consolidated of California.

Average depth.	Total ground worked.	Ground worked per day.	Area dredged.	Cost per cubic yard, cents.											
				Total expense.	Running expense.					Repairs.		General expenses.	Smelting and express.	Taxes.	Insurance.
					Labor.	Material.	Power.	Water.	Total.	Labor.	Material.				
Feet.	*Cu. yds.*	*Cu. yds.*	*Acres.*												
24	2,884,250	7,946	75.9584	3.93	0.59	0.06	0.70	0.08	1.43	0.19	1.75	0.41	0.03	0.08	0.04
21	1,816,849	5,005	54.0199	3.89	1.03	.18	.71	.19	2.11	.19	.95	.44	.03	.08	.09
37	1,617,392	4,443	27.0221	5.33	1.13	.15	.75	.14	2.17	.21	2.29	.48	.03	.09	.06
22	2,049,853	6,765	56.8314	2.67	.65	.05	.50	.07	1.27	.17	.59	.49	.03	.05	.07
54	1,802,644	4,966	20.5995	4.27	.77	.08	.84	.07	1.76	.33	1.56	.43	.03	.08	.08
40	1,396,212	3,846	21.6344	6.49	1.29	.28	1.06	.13	2.76	.28	2.64	.56	.03	.12	.10
60	1,469,431	4,048	14.8511	5.82	1.01	.14	1.25	.11	2.51	.52	1.99	.53	.07	.17	.03
67	2,053,568	5,657	19.0005	7.14	.84	.15	1.32	.10	2.41	.43	3.58	.57	.03	.10	.02
54	2,346,824	6,465	27.1771	5.25	.83	.18	1.41	.11	2.53	.26	1.74	.50	.03	.10	.09
37	2,273,942	6,264	38.0307	5.26	.88	.17	1.09	.12	2.26	.30	1.96	.51	.07	.14	.02
33	1,815,079	5,000	33.8020	2.50	.80	.09	.54	.....	1.43	.08	.47	.34	.02	.07	.09
35	1,509,316	4,158	26.7344	3.01	.87	.06	.60	.....	1.53	.13	.64	.41	.04	.15	.11
41	2,818,239	7,764	42.6772	3.31	.80	.15	.94	.....	1.89	.09	.77	.33	.03	.12	.08
34	19,710,985	5,520	355.0851	4.91	.87	.13	.96	.11	2.07	.28	1.88	.49	.03	.10	.06
38	6,142,634	5,641	103.2136	2.99	.82	.11	.73	.....	1.66	.10	.65	.35	.03	.11	.09
....	25,853,619	5,548	458.2987	4.45	.85	.13	.91	.08	1.97	.24	1.59	.45	.03	.10	.07

Working costs of gold dredging in California.[a]

Capacity of buckets.	Time in commission.		Working period for figures given.		Actual working time; hours during working period.[b]	Quantity handled.	Average depth of gravel.	Operating expenses, cents per cubic yard.							Remarks.
								Labor and material.	Electric power.	Water.	Repairs.	General.	Taxes and insurance.	Total expense.	
Cu. ft.	*Yrs.*	*Mo.*	*Yrs.*	*Mo.*		*Cu. yds.*	*Ft.*								
3	....		1		2,809	173,655	27.0	2.77	0.90	0.14	4.15	0.78	0.49	9.23	Difficult digging.[c]
3	5	9	1		7,216	458,882	26.9	2.03	.69	....	3.28	.63	.37	7.00	Working under favorable conditions.
3.5	7		1		....	395,316	35.0	2.83	1.53	.228	1.74	1.32	....	7.67	
3.5	6	6	1		7,344	461,882	35.0	2.85	1.48	.195	1.71	1.07	....	7.32	Compact gravel land subject to overflow.
4	9		1		7,057	484,387	20.6	1.83	.89	.31	2.58	.65	.26	6.52	Remodeled dredge, uneven bedrock, in places shallow.
5	6		1		....	481,184	25.0	3.28	1.46	.39	2.97	1.52	....	9.55	Difficult ground, in places cemented gravel.
5	2	5	1		....	635,146	27.0	3.14	1.45	....	2.40	1.28	.41	8.70	Difficult ground.
5	2		1		....	582,891	30.0	3.28	2.02	.32	2,59	1.37	1.00	9.60	Difficult digging.
5	5	6	1		7,344	615,009	25.0	3.06	1.42	.29	3.06	1.14	....	8.98	Do.
5	4	6	1		....	812,355	36.0	[d] 2.30	1.08	....	2.95	....	.35	6.65	Medium gravel with considerable clay, much brush on top soil.
5	3	5	1		6,798	1,148,480	25.5	.88	.52	.05	1.77	.25	.35	3.80	Loose gravel, heavy overburden of sandy loam.
5	2	5	1		6,790	1,148,802	29.9	.82	.49	.03	1.89	.25	.16	3.64	Do.
5	4	7	1		6,644	599,614	38.5	1.77	.92	.25	[e] 4.03	.47	.23	7.67	Difficult digging, working against 20-foot bank.
7		9⅓	1		5,088	838,885	35.0	1.19	.69	....	1.22	.26	.17	3.53	Difficult digging, gravel coarse, partly cemented.[f]
7	1		1		6,313	1,114,605	27.6	1.21	.62	.03	1.81	.29	.11	4.07	Compact gravel.
7	3	9	1		6,390	1,033,694	26.5	1.08	.64	.14	[g] 2.69	.34	.20	5.09	Compact gravel, heavy digging.
7	2	9	1		6,917	1,017,167	28.1	1.10	.65	.15	[h] 2.19	.28	.14	4.51	Do.
7	3		1		6,352	935,322	33.4	1.26	.85	.06	3.06	.31	.34	5.88	Compact gravel.
7	3		1		6,700	1,194,146	27.5	1.05	.58	....	2.78	.32	.37	5.10	Do.
7.5	2	11	2	11	13,464	3,458,229	27.9	.27	.41	....	1.50	.24	.24	[i] 4.42	Medium compact bench gravel.
7.5		9⅕		9⅕	5,582	944,879	28.9	.95	.58	....	1.30	.27	.39	3.55	Medium compact gravel with heavy overburden.
7.5	2	6	1		6,402	1,369,844	70.2	.99	.77	....	1.95	.45	....	4.16	Medium gravel overlain with hydraulic tailings.
7.5	2	6	1		6,900	1,281,351	67.8	1.09	.98	....	2.01	.45	....	4.53	Do.
8		6		6	3,162	583,927	42.5	1.69	.59	....	1.14	.28	.22	3.92	Light gravel, dredge working against 10-foot bank.
8		4¼		4¼	2,369	626,624	24.0	(j)	(j)	(j)	(j)	(j)	(j)	2.47	
9		5		5	....	580,310	51.0	(j)	(j)	(j)	(j)	(j)	(j)	4.98	Cemented gravel, difficult digging; 20-foot bank above water level.
13.5		8		8	4,478	1,803,201	19.0	1.02	.47	....	.60	.12	.09	2.30	Fine gravel, easy digging.

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[a] Janin, Charles, Present-day problems in California gold dredging: Trans. Am. Inst. Min. Eng., vol. 142, 1911, p. 873.

[b] Total possible time in year's work, 8,784 hours.

[c] A 7-foot dredge is now working this ground at a profit.

[d] Including general expense, management, etc.

[e] Heavy repair cost due to new tumbler, conveyor belt, repairs to digging ladder, screens, etc.

[f] This dredge successfully replaced an open-connected bucket dredge which could not handle ground at a profit.

[g] Replacing tumbler shafts, conveyor belt, and new screen included in repairs.

[h] New steel spud and screen in repairs.

[i] Depreciation charges included in total expense.

[j] Segregated costs not given.

Operating costs of dredges of Tonopah Placers Co.

Dredge No. 1.

Item.	Total cost.	Cost per cubic yard.
Labor	$18,296.69	$0.019
Supplies	8,445.09	.009
Power	14,262.36	.016
Total direct charges	41,004.14	.044
Superintendence	3,002.42	.003
Office	1,198.12	.001
Insurance	842.92	.001
Taxes	3,696.62	.004
General	670.02	.001
Company cottages	1,021.87	.001
Stable	115.39	......
Auto	557.21	.001
Total indirect charges	11,104.57	.012
Total	52,108.71	.056
Yardage, cubic yards	919,397	......
Cost per cubic yard	.056	......

Dredge No. 2.

Labor	17,218.08	.013
Supplies	9,782.57	.007
Power	17,501.72	.013
Total direct charges	44,502.37	.033
Superintendence	3,244.48	.002
Office	1,352.21	.001
Insurance	842.42	.001
Taxes	8,781.70	.006
General	615.99	.001
Company cottages	394.47	......
Stable	115.39	......
Auto	557.21	......
Total indirect charges	15,903.87	.012
Total	60,406.24	.045
Yardage, cubic yards	1,332,739	......
Cost per cubic yard	.045	......

Dredge No. 3.

Labor	20,838.80	.028
Supplies	16,630.97	.022
Power	17,925.62	.024
Total direct charges	55,395.39	.074
Superintendence	3,799.98	.005
Office	1,570.22	.002
Insurance	965.57	.001
Taxes	3,491.68	.005
General	619.64	.001

Legal	$138.85	......
Drilling	294.49	$0.001
Auto	557.27	.001
Stable	92.50	......
Total indirect charges	11,530.20	.016
Total	66,925.59	.090
Yardage, cubic yards	743,120	......
Cost per cubic yard	.090	......
Total cost all dredges	179,440.54	......
Total yardage	2,995,256	......
Average cost per cubic yard	0.059	......

Operating costs and miscellaneous data, dredges of Marysville Gold Dredging Co.

DREDGE NO. 3.

MISCELLANEOUS DATA.

Item.	1910	1911	1912	1913	1914
Working days	362	364	364	363	363
Working hours	8,688	8,736	8,736	8,712	8,712
Hours digging	6,106.39	6,050.59	6,493.00	7,322.34	7,373.51
Time digging, per cent	70	69	74	84	85
Area in acres	18.114	16.124	15.1373	15.7166	17.2700
Average depth, feet	60	62	66	65	66
Cubic yards handled	1,728,302	1,532,784	1,515,443	1,624,055	1,831,802
Cubic yards per day	4,774	4,211	4,163	4,474	5,046

COSTS PER CUBIC YARD.

Labor	$0.0182	$0.0161	$0.0166	$0.0160	$0.0149
Power	.0070	.0074	.0088	.0085	.0087
Supplies	.0235	.0382	.0330	.0282	.0222
Taxes and insurance	.0021	.0047	.0030	.0031	.0035
Miscellaneous	.0026	.0025	.0057	.0051	.0052
Total	.0534	.0689	.0671	.0609	.0545

DREDGE NO. 4.

MISCELLANEOUS DATA.

Item.	1911	1912	1913	1914
Working days	239	364	363	363
Working hours	5,736	8,736	8,712	8,712
Hours digging	4,428.06	7,075.10	6,901.08	7,415.33
Time digging, per cent	77	81	79	85
Area in acres	11.6645	15.5522	19.7450	22.5583
Average depth, feet	59	61	58	61
Cubic yards handled	1,058,797	1,453,428	1,808,001	2,112,654
Cubic yards per day	4,430	4,000	4,980	5,820

COSTS PER CUBIC YARD.

Labor	$0.0156	$0.0176	$0.0142	$0.0125
Power	.0081	.0102	.0083	.0078
Supplies	.0191	.0318	.0191	.0189
Taxes and insurance	.0041	.0025	.0030	.0032
Miscellaneous	.0023	.0069	.0045	.0046
Total	.0492	.0690	.0491	.0470

Operating costs of dredges of Oro Electric Corporation.

OROVILLE DREDGES.

Year.	Dredges operating.	Cubic yards dredged.	Days elapsed.	Average hours running time per day.	Average cubic yards per month per dredge.	Average cubic yards per day per dredge.	Average cubic yards per hour per dredge.	Average cost per month per dredge.	Cost per hour of running time.	Cost per cubic yard, in cents.				
										Maintenance.	Operating.	General.	Insurance and taxes.	Total.
1903.	0.25	196,479	214	5.39	19,560	652	120.9	$836	$5.17	......	......	......	......	4.27
1904.	1.83	1,008,090	824	10.20	32,850	1,095	107.4	2,365	7.73	......	......	......	......	7.20
1905.	3.00	2,210,609	1,095	16.74	60,700	2,020	120.7	3,942	7.85	......	......	......	......	6.50
1906.	3.67	3,131,020	1,309	19.10	69,060	2,302	120.5	3,360	5.86	......	......	......	......	4.78
1907.	4.75	3,755,115	1,582	17.02	69,300	2,310	135.7	3,670	7.19	1.72	2.95	0.54	0.086	5.31
1908.	4.58	4,572,573	1,830	16.04	74,880	2,496	155.6	4,192	8.71	2.24	2.46	0.43	0.476	5.61
1909.	5.00	4,004,352	1,825	16.53	66,060	2,202	133.2	4,043	8.15	2.69	2.64	0.44	0.361	6.13
1910.	5.00	4,308,476	1,825	18.08	70,890	2,363	130.6	3,539	6.52	1.65	2.65	0.44	0.263	5.00
1911.	4.58	3,646,695	1,641	18.74	65,520	2,184	116.6	3,644	6.48	1.79	2.95	0.44	0.374	5.56
1912.	4.00	2,225,948	1,464	17.63	46,374	1,520	86.2	3,687	6.85	2.66	4.25	0.43	0.606	7.95
1913[a]	4.00	3,465,711	1,460	18.71	72,202	2,374	126.9	3,745	6.57	1.58	2.92	0.34	0.347	5.19
1914.	3.76	3,303,647	1,374	19.08	73,218	2,404	126.0	3,057	5.26	0.80	2.73	0.30	0.352	4.18
CAMANCHE DREDGE (9 cubic feet).														
1913.	0.88	1,812,533	322	20.02	170,351	5,629	281.1	5,550	9.16	1.02	1.79	0.31	0.140	3.26
1914.	1.00	1,704,044	365	19.38	142,004	4,669	239.4	5,668	9.56	1.23	2.16	0.31	0.289	3.99

[a] March, 1913, one dredge was converted from 5 to 6 cubic feet capacity.

RECOVERY OF GOLD COMPARED TO CONTENT INDICATED BY PROSPECTING.

What percentage of the gold is recovered by a dredge, whether the actual content of the material dredged or the yield indicated by prospecting is considered, has always been an interesting question. It is generally acknowledged that the modern dredge with its improved washing plant and large area of gold-saving tables recovers, except where considerable sticky clay or where "rusty" gold is present, a high proportion of the gold in the material washed. The proportion of gold recovered from the amount estimated to be in the ground is a different matter, and requires the consideration of several factors not directly connected with the metallurgy of gold dredging. Obviously, to determine the difference between the actual gold content of the ground and the recovery on the dredge, it is necessary to know: (1) the gold content of the ground previous to dredging, or (2) the loss in the tailing.

Too much dependence on a vendor's statement of past production or his estimate for the future accounts for many a wide discrepancy between expectation and fulfillment, and the same may be said of careless or inexperienced sampling or, sometimes, no sampling whatever. In the early days of dredging, inexperience in the use of drills for sampling placer ground often led to over estimation of the gold content; this error accounts in part for the first dredges recovering a low proportion of the gold indicated by drill sampling. Experience has minimized this danger, and the results obtained over a period of years have shown that there is little reason to-day for a wide discrepancy between the actual recovery and the content estimated by prospecting, provided the prospecting was done by an experienced engineer with sufficient funds and time at his disposal.

The thorough sampling of a placer, as described in the chapter on prospecting, is by no means the simple matter the uninitiated often believe it to be. Much depends on the care taken and the experience of the engineer in charge.

If great care has been exercised in sampling, every detail being done properly, each sample will represent as nearly as possible the gold content per cubic yard at the place sampled. Although each hole may be taken as indicative of the content of the ground where it is sunk, the proper placing of the holes is of importance. The proper location of holes and the interval at which samples should be taken can be determined only after a study of the ground. There is no hard and fast rule to follow. Some areas have been considered sufficiently prospected with one drill hole to 10 acres; on other areas the engineer has not been fully satisfied with one hole to one acre.

METHODS OF PROSPECTING.

Preliminary holes are generally sunk and carefully sampled to determine whether the gold occurs in wide or in narrow channels, and whether its distribution is fairly even or in irregular pay streaks. Holes badly spaced with too many in the richer part of the ground would yield too high an average content. Ground that shows a spotted or irregular gold content will obviously need more tests than ground yielding fairly uniform samples. Where sampling discloses a channel or narrow belt having a higher mineral content than that of the whole area, the outlines of the channel or belt should be determined as closely as the purpose of the examination warrants. Such sampling is needed at French Gulch near Breckenridge, Colo. where the principal gold content is in a narrow channel, and ground outside the limits of the channel is too poor for profitable dredging.

Sometimes it is feasible to segregate certain parts of a property and eliminate low-grade areas. When a tract is large and the time for examination is limited, tests to confirm the samples indicated by preliminary prospecting may be concentrated on a piece of ground large enough to determine the advisability of purchasing the property and installing at least one dredge. From these statements it is clear that the accuracy with which the average mineral content of placer ground is indicated by an examination depends on the care taken in sampling, the number of samples taken, and the situation of the holes with reference to the character of the deposit. Everything hinges on the experience and judgment of the man in charge.

FACTORS USED IN CALCULATIONS.

In estimating the probable recovery from dredging ground most engineers, after determining the average content indicated by prospecting, take a certain proportion, generally 75 or 80 per cent, of this indicated content as recoverable by the dredge. These figures are based on actual tests made to determine the dredge recovery, and also on the percentages of the prospecting content recovered in dredging certain carefully prospected areas.

Some engineers, when sampling with drills, use a different factor in their calculations than that generally employed; that is, they reduce the metal content per cubic yard of the place sampled by an amount that, in accord with their experience, represents recovery. Use of this factor, practically a combination of drill factor and recovery estimates, discounts the content that otherwise would have been assumed from the regular drill factor. For instance, in place of the usual factor 27, a new factor, 33⅓ per cent, is used. Engineers following such a practice should mention it clearly in their reports. Failure to do so might cause the future of a property to be discounted further by persons to whom the report is presented, if these persons are not conversant with the methods employed in computation.

Attempts to confirm the gold indications of individual drill holes by sinking shafts around the drill holes, or sinking other drill holes near the first, generally yield widely different results. The gold is not uniformly distributed throughout the gravel, so that the results from single drill holes do not indicate of average content. There are rare instances on record of the dredge recovery exceeding estimates, not only for the drill showing on certain small areas between clean-up periods, but also for the total area dredged. Thus in a small area dredged by Natoma No. 4, 120 per cent of the yardage, and 115 per cent of the estimated recovery were obtained. In most discrepancies of the kind it will be found that ordinary methods of valuation were discounted in preparing the estimates.

The following reference to the Pato property, taken from the report of the Oroville Dredging Co., dated September 30, 1911, shows the prevailing practice in discounting the prospecting indications. "The original estimates by the engineers give us an area containing 13,637,347 cubic yards of material, yielding gross $3,202,986, there having been deducted from the values recovered by protecting tests 25 per cent for variation, possible losses, etc."

RECOVERY FROM DIFFERENT PARTS OF TABLE.

Figure 22, representing the gold-saving table of a 6-foot close-connected bucket dredge, furnished by W. H. James of Oroville, shows the percentage of the total gold recovered from different parts of the table. It is based on careful data kept at each clean-up for a period of three years.

The total table area is 1,457 square feet; 89 per cent of the gold recovered is arrested on 292 square feet, as shown on those portions of the table marked 1. Including the recovery from the save-all, 92.2 per cent of the gold was obtained from 332 square feet; 4.68 per cent more was recovered from 360 square feet additional. Thus 96.9 per cent of the total gold recovered on this dredge came from 692 square feet of table area. The remaining 3.1 per cent was obtained on the additional 765 feet. Although the total table area on this dredge was 1,457 square feet, that on a dredge having the same bucket capacity, but designed for ground containing much clay and difficult to wash, was increased to 3,800 square feet.

Table area varies with the ideas of designers and with the conditions to be encountered. From the first dredges with only a few square feet of table area to the modern 15-foot dredge with 8,000 square feet is a long step, as is also the increase of capacity from 10 cubic yards or less per hour of actual mining time for early dredges to 500 cubic yards per hour for months at a stretch for some modern boats.

Figure 22 is interesting as showing how large a proportion of the gold is recovered on the upper part of the table, the bulk of all the gold recovered being actually taken from the first 3 feet at the top end. It is advisable, however, to have as large a table area as space will permit, even though it is not often necessary to clean up the lower end. Tests with the Neill jig, described in the chapter on metallurgy, indicate the advisability, on some boats, of supplementing

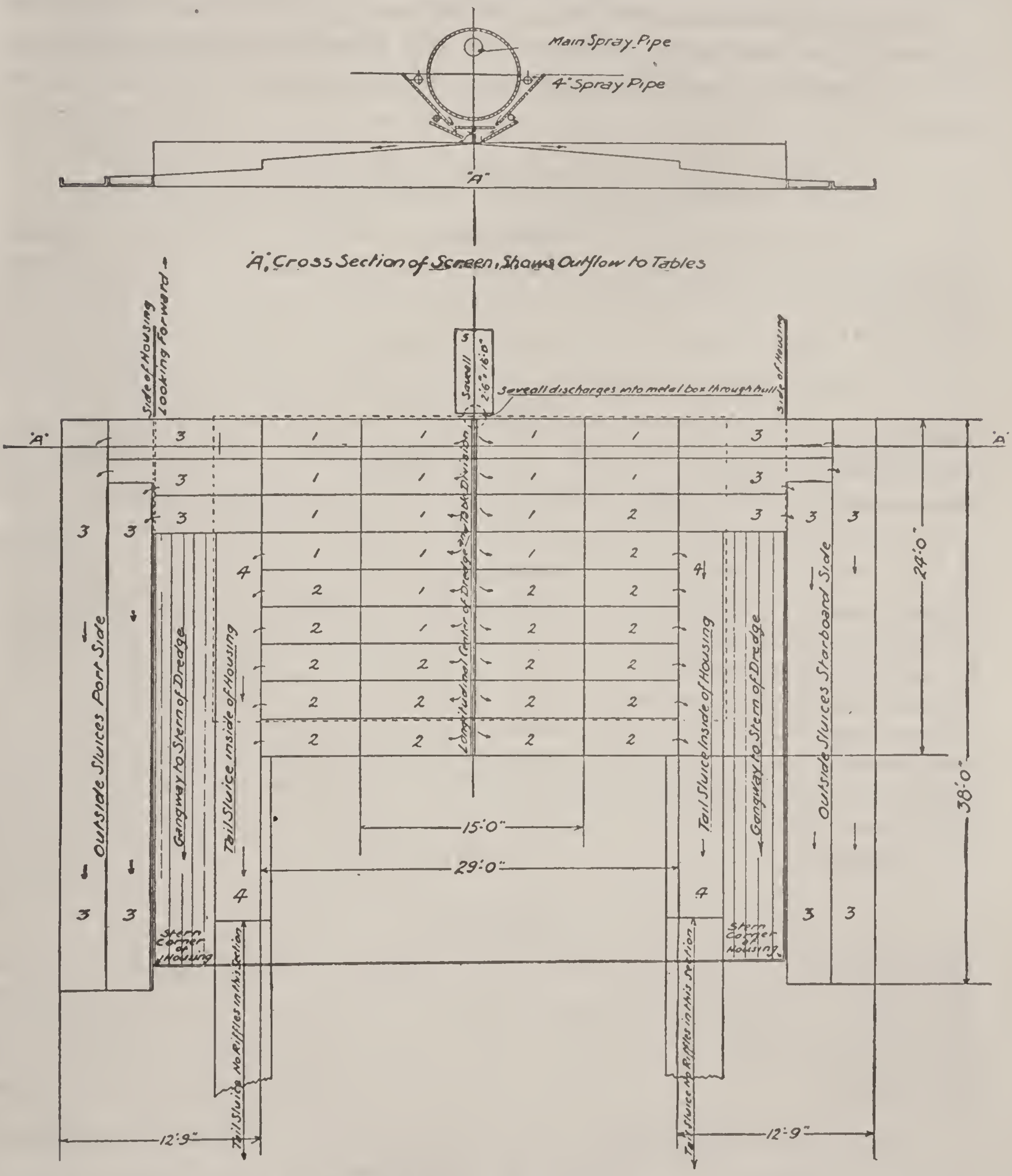
Main Spray Pipe
4" Spray Pipe
"A"
"A", Cross Section of Screen, Shows Outflow to Tables
Side of Housing
Looking Forward
Saveall
5
2'6" x 16'0"
Saveall discharges into metal box through hull
Side of Housing
"A"
"A"
Longitudinal Center of Dredge & Table Division
Outside Sluices Port Side
Gangway to Stern of Dredge
Tail Sluice inside of Housing
Tail Sluice inside of Housing
Gangway to Stern of Dredge
Outside Sluices Starboard Side
1
2
3
4
24'0"
38'0"
15'0"
29'0"
Stern Corner of Housing
Stern Corner of Housing
Tail Sluice No Riffles in this Section
Tail Sluice No Riffles in this Section
12'9"
12'9"

FIGURE 22.—Table plan of gold-saving area on dredge equipped with revolving screen, showing distribution of flow and recovery from segregated areas.

the gold-saving tables with other devices. Few operators have ever complained of having too large a table area on their dredge, and many would add to the existing tables if space permitted. Investigation of the tables of a dredge will generally show some loss of quicksilver, and consequently of gold, over the tail sluice in addition to the fine gold carried off in suspension.

From the amalgam returns, the percentages recovered from the different gold-saving appliances on the Conrey dredges at Ruby, Mont., from the start of their work to December 31, 1913, are as under:

Proportion of total recovery from gold-saving devices on Conrey dredges.

	Per cent.
Dredge No. 1.	
All 20 tables	92. 54
Tail sluices	3. 31
Well save-all	4. 15
	100. 00
Dredge No. 2.	
All 20 tables	93. 46
Tail sluices	3. 46
Well save-all	3. 08
	100. 00
Dredge No. 3.	
Upper flume	55. 66
Lower flume, first section	27. 11
Lower flume, second section	6. 72
Lower flume, below undercurrent	. 49
Undercurrent	1. 33
Well save-all	5. 01
Tables	3. 01
Tail sluices	. 31
Miscellaneous	. 36
	100. 00
Dredge No. 4.	
First 16 tables	87. 90
Last 8 tables	1. 96
Tail sluices	6. 81
Undercurrent	. 57
Well save-all	2. 76
	100. 00

GOLD IN TAILINGS.

To sample the dredge tailing is difficult. To be trustworthy, any test must extend over a considerable period. Practically all such tests have indicated an appreciable loss, but most of the tests did not, so far as the author knows, determine the gold recovery with any accuracy. Even the extensive tests on Natoma No. 7 dredge prior to installing the Neill jig, although they demonstrated a high loss of gold, did not establish this with any exactness. Most statements of exact percentages of loss of gold have proved, upon investigation, to be merely guesses made by the dredge master. In connection with one of the most thorough tests made for this purpose, at Oroville, a small bucket-elevator was used. The tailing examined was taken by means of perforations in the bottom of the tailing sluices; it passed to a small wooden sump, in which the lower sprocket wheel of the elevator was mounted. The material was somewhat concentrated, because of the method of collection, but this could not be avoided. The speed of the elevator was arranged so that approximately 2 cubic yards of material would be collected during the shift. This collected tailing afterward was carefully washed through a rocker, and panned. All the tests showed an appreciable loss of quicksilver and amalgam, but no definite figure of loss, compared with the total gold content of the material handled, was obtained.

In general, it may be said that the loss of gold in the tailing is less than the amount lost in front of the boat, both from bucket spill not caught in the hopper and save-all, and from failure properly to clean bedrock. It is estimated by those competent to judge that the average recovery of the actual gold content, while varying with different operating conditions and efficiency of labor, is not less than 90 per cent under normal conditions, and sometimes it may be considerably more. Compare this with the following extract from the Canadian Mining Review of January 30, 1903, in reference to the old Cobeldick dredge at Lytton:

> The director who was sent out from England to investigate for the company the working of the dredge made the discovery that of the gold dredged up from the bottom less than 10 per cent was recovered on the tables, the remaining 90 per cent going off again with the tailing, although the saving appliances on this machine were about the most complete of any in British Columbia.

It would be interesting to learn what tests the director made to arrive at this conclusion. Probably the recovery was only 10 per cent of the estimated amount, and the blame was put on the gold-saving appliances instead of where it rightly belonged. The question was, perhaps, not so much one of poor recovery as of an over-estimate of gold content.

EXAMPLES OF RECOVERIES BY DIFFERENT DREDGES.

In a careful test to determine dredge recovery, 22 holes were sunk on a measured area. Results were from 3 to 52 cents a cubic yard, and averaged 19 cents; the regular drilling factor was used in the calculations without any discount for recovery. In working, the recovery was, as nearly as the author can remember, 17 cents a cubic yard, or about 90 per cent of the content shown by averaging the samples obtained in drilling. In another test 17 holes were put down in front of a dredge, the content of the area closely calculated according to drill methods then employed, and the total content discounted 20 per cent. The dredge recovered 99 per cent of the total amount estimated to be in the ground. Those companies in California that get the highest recovery of the estimated content are those that discounted the prospecting indications by 20 or 25 per cent. To have much value, the recovery percentage based on prospecting must be founded on bank measurement of the yardage handled—not on the size of the dredge bucket and the running time of the dredge.

The following results, obtained by a California company working one dredge only, are of considerable interest, because the company kept careful records from the beginning.

Results obtained by a California company.

Acres dredged	144
Cubic yards dredged	6,980,800
Gold content per cubic yard as shown by drill holes, before discounting ....cents..	21.05
Actual recovery ....do....	12.92
Recovery of amount indicated by drill holes ....per cent..	61.38
Drill holes to area (approximately 1 hole to 3 acres)	47

RESULTS FROM A SMALL AREA.

Acres dredged	40
Cubic yards dredged	1,232,868
Prospecting content (cents per cubic yard)	17.18
Recovery (cents per cubic yard)	15.35
Percentage of recovery	89.34
Six holes to 40 acres.	

The results obtained by the dredges of the Natomas Consolidated and the Yuba Consolidated are of especial interest on account of the magnitude of the operations. The dredges of the Natomas Consolidated from the beginning of operations to December 31, 1915, handled approximately 150,000,000 cubic yards. The following excerpt, in regard to the recovery, is taken from C. M. Rolker's report on the Natomas Consolidated in 1909:

> The writer wishes to emphasize that according to the statement of the general manager, Newton Cleaveland, the values of the different bore tests, which are marked on the company's maps, are the original values calculated, reduced by 15 per cent to make them representative of recovery.

In 1913, Frank Griffin, the president of the Natomas Company, stated that the recovery made by the dredges was 15 per cent less than the engineers estimated. This would be about 72 per cent of the yield indicated by the drill holes previous to the 15 per cent reduction, mentioned in Mr. Rolker's report. It should be noted, however, that all of the ground has not been dredged, and it is only when the property is worked out that the actual recovery of the gold assumed to be in the ground will be ascertained.

The dredges of the Yuba Consolidated had handled over 135,000,000 cubic yards to February, 1915. The actual recovery as compared with that estimated is not yet ascertainable, but, according to a large shareholder, it is about 75 per cent of that indicated by prospecting. The percentage of recovery is difficult to determine as a dredge will at times work for months without encountering a drill hole and until the property is finally worked out, no figures for recovery as compared to first estimates, can logically be made. Final figures will be interesting, as the company has never been advertised, the stock is privately held, and the profits are said to exceed those of every other gold-dredging company.

The California dredges of the Oroville Dredging (Ltd.) have handled over 50,000,000 yards to January, 1916. At the time when the property was floated in London there were estimated to be 38,500,000 yards with an average content of about 17.2 cents, and the Bear River property was stated to contain an average of 12 cents. The prospecting results as indicated by drill holes on some of this ground had been discounted previous to the consolidation, but seeming not on the larger part of the property. The average recovery from dredging 44,800,000 cubic yards, exclusive of Bear River gravels, was 11.17 cents per yard, indicating a recovery of approximately 65 per cent of what was expected.

The Bear River area was even more disappointing. The average from 1,832,585 cubic yards during three years was 5.26 cents, or about 44 per cent of the estimated content. The recovery made by the Pato dredge, operating on the holdings of this company in Colombia, have been the subject of considerable comment, as the yield has been at times far in excess of the average gold content as estimated for the entire area. It is clearly understood by those with experience in such matters that the recovery made by a dredge over short periods often varies greatly. Discrepancies of recovery over such periods compared with the average gold content assumed for the whole area are no more significant than high gold content in a few drill holes as compared with the average of the property, other than perhaps to indicate a richer spot or "channel" in the area. In other words, because the recovery from a property for a short period in the beginning of operations is higher than the average for such property, there is no reason to expect the whole area to exceed the assumed average.

The results from Pato show that during two seasons 2,500,000 cubic yards, yielding an average of 43.8 cents, has been dug in an area of 92 acres, which were estimated to contain a total of 3,780,000 cubic yards having an average content of 40.4 cents. Although the recovery of the dredge per cubic yard for the first 750,000 yards from the "tested" area was 54 per cent more than the estimated content as indicated by the drill for the whole 92 acres, and occasioned considerable comment at the time, it meant only that the dredge was operating in the richer parts of the "tested area." No comparison of recovery as compared with average prospecting content could rightly be made on a small section of a large tract.

It has been stated [a] by the Colorado Gold Dredging Co., operating in Colorado that the recovery from its dredge under the best conditions averaged about 80 per cent of the gold previously estimated to be in the gravel. Other Colorado companies, however, did not succeed in obtaining as high a recovery. In fact, a number of the earlier dredging ventures in Colorado proved unprofitable, though doubtless the gold content assumed to be in the ground was sufficient to justify the venture.

Although most of the figures mentioned have been based on prospecting done under the direction of careful engineers, there are numerous instances, particularly of early drillings, where the recovery made by the dredge has been 40 per cent or less of the assumed average gold content of the ground. The dredges of one California company handled 8,156,571 cubic yards and obtained a recovery of 40.6 per cent of the gold indicated by prospecting.

FAULTY PROSPECTING A CAUSE OF SEEMINGLY POOR RECOVERY.

At times a poor recovery is made by a dredge on account of careless or inexperienced work during the examination. Although bore holes indicated 5 to 13 grains per cubic yard, one of the dredging companies in West Africa never recovered more than 2 grains per yard. Ransome [b] mentions that methods employed in drilling and calculating gold content are not always uniform and cites an instance showing the comparative results of two methods used by the Colorado Gold Dredging Co. to estimate the value of ground tested by drill holes. By one method of computing, using the diameter of the drilling shoe, a content of 4.6 cents a yard was indicated, and by the other, figuring the volume of the material pumped from the same hole, a gold content of 25.4 cents a yard was indicated.

Mention may also be made of a property near Nome in Alaska, where the value of the gold recovered was less than 20 cents per yard from gravel that was claimed would average more than $1. It may be said, however, that this instance represented not so much a poor recovery as fraudulent representation. What was needed here, and would be valuable elsewhere under like conditions, was not so much a factor representing dredge recovery as one to discount vendor's statements.

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[a] Bradford, A. H., and Curtis, R. P., Dredging at Breckenridge, Colo.: Min. and Sci. Press, vol. 99, Sept. 11, 1909, pp. 361-366.

[b] Ransome, F. L., Geology and ore deposits of the Breckenridge district, Colorado: Prof. Paper 75, U. S. Geol. Survey, 1911, p. 181.

RECOVERIES FROM REDREDGED GROUND.

Some ground that was redredged in California in moving boats to new positions returned 1 to 3 cents a yard—not sufficient to pay operating expenses, but from one area a dredge recovered gold to the value of 10 cents a yard when working through tailing. An investigation resulted, and a considerable part of the tailing was redredged at a profit. Imperfect washing of the material caused by the poor arrangement of the washing plant and the gold-saving tables, which were replaced with more modern equipment, was the cause of the poor recovery, and perhaps the desire to make yardage records with the earlier dredge was also an essential factor. Elsewhere, from gravel containing much clay or cemented gravel, the recovery of gold may be low and the tailing show sufficient gold to make redredging practical. In some parts of the Oroville field clay dredged six years ago and since exposed to the weather has not yet disintegrated.

In northern California the recovery recently made by a dredge was 15 cents a yard when working through the tailing from a former dredge. The reason was that the earlier dredge had not reached bedrock. The first dredge did not prove a financial success but the second has been a profitable venture from the beginning.

At Jenny Lind, Calaveras County, as high as 7 cents a yard has been recovered in redredging ground. On the other hand, Dredge No. 3, Feather River, of the Natomas Consolidated Co., dug through 500,000 cubic yards of tailing from the old Cherokee track, also digging about 100,000 yards of virgin ground during the same run. The recovery from the 600,000 yards handled averaged only 1½ cents a yard, and the greater part of this recovery must have been from the new ground, indicating that the tailing was practically barren.

The Pennsylvania dredge at Oroville in moving to a new location cut through over half a mile of tailing and recovered about $450, no attempt being made to dig deeper than necessary for flotation. Other dredges in the Oroville district in working through tailings from former operations have recovered from 1 cent to as high as 10 cents a cubic yard over short periods, the high recovery representing that of a dredge digging near bedrock ridges that had been left between cuts in former dredging.

A washing plant was built by the Natomas Consolidated to handle the fine material screened from the rock sent to the rock-crushing plant. All material smaller than ⅛ inch was sent to this plant, which had riffled sluices and tables. The recovery when the plant was first tried indicated about 3⅓ cents a yard for the material sent to the tables and about ⅓ cent for the total material handled. The upper 2 feet of fine material left after the coarser rock is taken to the rock-crushing plant has given rather high results at times, and it may be possible to redredge some material at a profit after the rock has been removed and by reclaimimg the land behind the dredge add to the profit of the operator.

At Ruby, Mont., although the entire property has never been systematically prospected as a whole according to modern methods, small areas have been more or less closely prospected by drill holes and afterwards worked. The percentage of recovery by the dredges has generally averaged about 80 per cent of the probable recovery indicated by prospecting, the ordinary drill factor being used for valuing the gold content of each hole. The percentage of the actual gold content recovered by the dredge is of course a different matter. Drill holes in the tailing of the present dredge have shown little gold and when it was necessary to dig through any of this tailing in moving the dredge to a new position the recovery has been about 0.54 cent a cubic yard in a total of approximately 125,000 yards thus reworked, indicating a close recovery of the gold content on the first dredging where the ground is easily washed. In May, 1915, the No. 3 dredge dug through a quantity of tailings from the old steam dredges and made a recovery of 2.05 to 3.73 cents a cubic yard, demonstrating that the earlier dredges were not as efficient gold savers as the modern ones. The figures indicate a loss with the formerly used flume dredge varying from 10 to perhaps 20 per cent of the actual content.

RECOVERY OF GOLD AT OROVILLE, CAL.

The following table prepared by one of the most careful California operators is of considerable interest, and is based upon the following data: In dredging the property, a clean-up was made at the end of 15 days' dredging. It was found frequently that there had been a prospect drill hole on the area dredged during a 15-day period. If the hole was near the center of the area the recovery was compared with the content indicated by the drill hole. If the hole was reached at or near the end of the 15-day period, an average of two such periods would be used as the recovery figure for comparison with the content indicated by prospecting.

Comparative drill-sheet values and actual dredging results at Oroville, Cal.[a]

[Arranged in order of highest drill-sheet results and reduced to percentages of recovery.]

FIGURES OF ENGINEER "A".

Designation of drill hole.	Drill-sheet value.	Actual dredge value.	Per cent recovered.
	Cents per cubic yard.	*Cents per cubic yard.*	
F	59.80	20.72	34.60
G	48.51	26.50	54.60
C	42.30	19.56	46.30
E	33.53	13.36	39.80
U	32.79	15.56	47.50
I	26.20	4.60	17.50
L	23.81	14.83	62.30
A	21.35	9.25	43.30
N	21.23	10.37	48.80
V	20.80	14.00	67.30
M	20.25	11.17	55.16
K	18.52	14.80	80.00
J	15.47	7.97	51.50
A_2	14.30	13.28	92.90
D	11.14	17.00	152.60
B_2	10.76	5.77	53.60
T	8.31	13.82	166.30
B	7.53	10.40	138.00
Y	6.02	19.35	321.00
Z	2.58	15.50	600.00
X	1.85	7.25	390.00

FIGURES OF ENGINEER "B."

6	77.18	23.58	30.00
2	41.28	14.00	35.00
14	29.75	10.05	33.78
5	26.14	12.00	46.00
4	25.76	10.25	40.00
3	22.12	12.93	59.00
7	20.61	11.50	51.00
8	6.24	5.24	84.00
16	4.47	4.60	103.00

FIGURES OF ENGINEER "C."

F [b]	60.40	20.72	34.30
6 [b]	54.54	23.58	43.20
W	36.78	28.00	76.10
15	34.22	16.86	49.00
h	33.53	30.00	90.00
a	27.72	15.24	54.10
C [b]	27.10	19.56	72.00
j	22.24	17.80	80.00
b	21.15	8.30	39.20
i	18.88	15.44	82.39
d	18.37	20.60	112.20
5 [b]	17.83	12.00	67.30
e	17.49	19.16	109.50

[a] Janin, Charles, Recovery of gold in dredging: Min. and Sci. Press, vol. 109, 1914, p. 717.

[b] Check hole to verify results of hole previously drilled having similar designation. (See above.)

The data show that in the field worked by this dredge at least the record of a single drill hole can not be regarded as representative of the actual gold content of the surrounding area. This has been the experience of other operators. In one field, where the drill hole indicated 2.58 cents a yard, as shown on the table, the dredge recovered 15.5 cents, or 600 per cent, of the content given by the drill hole; in another field, from ground where the drill showed a content of 59.8

cents, the dredge recovered 20.7 cents, or 34.6 per cent; and in a third, where 26.2 cents was indicated, only 4.6 cents, or 17.5 per cent, was recovered.

Any estimate to be at all trustworthy can be made only from the average of the content indicated from a number of drill holes that have been intelligently placed and carefully sampled. If the work of prospecting has been done efficiently, an estimate of the gold content of a property can be made with some degree of accuracy, but if the work has been carelessly performed by unreliable or inexperienced engineers, the results obtained are so worthless as to render any refinement or calculation merely ridiculous.

RECOVERY AT SNELLING, CAL.

The following example of recovery as compared to prospecting is extracted from a letter of J. W. Neill: [a]

> The tract in question, belonging to the Yosemite and Mining Co. at Snelling and comprising 30 acres, lay on the opposite side of the river from the general mass of the ground and was left until the last for this reason. It was of importance to know whether this area was profitable, as, if so, we would cross the river, and dismantle the boat close to the county road, thus saving much distance in hauling. Holes were drilled as per the first map, 400 feet apart up and down the stream, by 200 feet across the flow, making 20 holes in all. The map shows the depth and the value as shown by the samples. The area was in part badly overgrown with brush, and in part in a slough, so we had to drill a few holes rather out of alignment. The top soil, consisting of sandy loam, averaged about 6 feet deep; below this was gravel consisting of small cobbles and sand. The bedrock was soft lava ash.
>
> The average of these holes, figured in the regular way, is 5.29 cents per yard. We use the "0.01 formula" in our estimates, and for the probable dredge yield deduct from this 20 per cent for losses. This would make the probable value of the area only 4.32 cents. Moreover, in this area there are three high holes—25 cents, 14 cents, and 20 cents. If we cut these in half, as being freaks, we get a value for the area of 3.63 cents and 2.90 cents dredge yield.
>
> To see if there was not a possible area of sufficient value to warrant dredging a second series of holes was put down, as in map No. 2, and for this prospecting an average of 4 cents per yard, drill yield, or 3.3 cents dredge yield, was obtained, including two freaks of 35 cents and 13 cents. By cutting these in two a drill value of only 3.27 cents and a dredge value of 2.61 cents is indicated. These two samplings, therefore, checked within a small fraction of a cent against each other, and the total average of both, which is for 45 holes in 30 acres, 4.57 cents, including all holes, or 3.42 cents if half the freak holes be deducted. The comparison would therefore be: First 20 holes, 3.63 cents; second 25, 3.27 cents; of all 45, 3.42 cents. After much discussion we selected a piece of the ground, crossed the river, and dredged, as shown in map No. 3, seven acres. This dredged area covered 14 drilling holes whose average value was 7.75 cents, or if we deduct as before the half of three high ones—20 cents, 33 cents, and 13 cents—we get 5.55 cents as the drill value, or an expected dredge yield of 4.44 cents. The records of our bullion product show for these three months an average yield of 6.74 cents per yard. In other words, we saved by the dredge a little more than 80 per cent of the drill returns of all holes, including the freaks.

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[a] Neill, J. W., Letter relative to methods of testing placer gravels: Min. and Sci. Press, vol. 109, Aug. 8, 1914, pp. 221-222

POWER PLANTS FOR DREDGE INSTALLATIONS IN ISOLATED DISTRICTS.

In the construction and operation of dredges in out-of-the-way places the power factor is one of the highly important problems. Where electricity is available it is the most nearly ideal power for dredge operation, but often electricity is either not available or can be developed only at an excessive cost. For the operation of only one dredge, obviously great expense can not be undertaken in power lines and electric-power installation, but if a company is building a number of dredges or if several companies contemplate dredge operation in a new field a central power plant to generate electricity will as a rule prove economical. Should electric power be neither available nor feasible then some other form of power must be considered.

The first steam dredges were excessive consumers of fuel, and operating costs were high in comparison to those when electricity became available for use on the same dredge. An example of the reduction in the power cost in dredges is that shown on the operation at Ruby, Mont. With the cost of coal at $4.90 a ton the fuel costs of the two Conrey, 7½-foot and 10-foot, steam dredges during 1904 to 1906 was 3.51 and 3.07 cents a cubic yard. Although these were old-type and inefficient machines, the figures cited illustrate the high power cost under steam operation even with a low fuel cost. The cost of power and fuel for the present Ruby dredges averages less than 1½ cents a cubic yard.

Practically all dredges now operating in the United States use electricity. In Alaska where the single-dredge installation is the rule and on the Seward peninsula where upward of 100 dredges have been built, 40 of which were either operating or under construction in 1913, only two used electricity. In this field and in the Philippines the American dredge constructor has had the opportunity to exercise his ingenuity and here most experiments have been made and the greatest development has taken place in the power arrangement of the self-contained dredge. In 1912, the writer [a] published an article on dredging in the Seward Peninsula which included a table giving the fuel used and the quantity consumed on most of the dredges in that field. Following is an abstract:

Fuel consumption on dredges in the Seward Peninsula, 1912.

Name of dredge.	Size of buckets.	Average daily capacity.	Horsepower of dredge.	Fuel used each day.	Quantity.	Cost of fuel per unit.	Approximate cost of fuel per cubic yard.	Operating conditions.
	Cubic feet.	*Cubic yards.*						
Solomon	3	1,800	160	Coal	7 tons	$27.00	10.5	Average.
Three Friends	5	2,000	230	Coal	12 to 17 tons	22.00	15.4	Difficult.
Wild Goose	3½	1,800	140	Distillate	300 gallons	0.25	4.2	Average.
Plein	3½	800	130	Crude oil	14 barrels	3.00	5.2	Some frozen ground.
Blue Goose	5	800	160	Wood	7 to 12 cords	12.75	12.7	Average.

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[a] Janin, Charles, Gold dredging on the Seward Peninsula: Min. and Sci. Press, vol. 105, Sept. 28, 1912, p. 396.

Since the table above was made, in 1912, there have been a number of changes made on these dredges, and two have shut down. The results indicated, although representative of work at the time specified, can not be considered as representative of the best present-day practice. It must also be noted that the fuel costs do not include labor, etc.

The use of gas engines demonstrated that a great saving could be made in the cost of fuel on dredges using coal, wood or crude oil when transportation of such fuel from the seaport was necessary, and continued experimentation demonstrated that still further saving could be made by improved methods.

EXAMPLE OF FUEL SAVING BY REDESIGNING POWER PLANT.

An interesting example of reduction of fuel costs is furnished by the operation of the Three Friends or Seward dredge. This dredge was built in 1908, and for reasons satisfactory to those interested in the company, was equipped with slide-valve engines and locomotive boilers. The coal consumption varied from 12 to 17 tons a day and the cost of coal delivered to the dredge was $22 a ton; with 14 tons at $22 as the average consumption, the fuel cost was $308 a day. The average daily capacity of the dredge was 2,000 cubic yards, so that the fuel cost was 15.4 cents a cubic yard. In the latter part of 1912 oil burners were installed on the dredge and fuel oil costing about $3 a barrel was used as fuel. This change, although reducing the fuel cost somewhat, did not give the results the management anticipated, some days as high as 60 barrels of oil being consumed. The writer pointed out [a] the folly of trying to do anything with the antiquated equipment, but it was not until 1914 that any radical change was attempted. The entire plant was redesigned. Semi-Diesel engines were used to generate electric power at a shore plant and electric motors were used on the dredge. The old machinery was consigned to the rock pile.

Figures of operating cost of the new plant, furnished by the management, are as follows:

Operating cost of Three Friends Dredge for 24 hours, 1914.

200 gallons of distillate, at $0.18 at power plant	$36.00
3 winchmen, at $7	21.00
3 oilers, at $5	15.00
3 engineers, at $8	24.00
2 roustabouts, at $5	10.00
1 dredgemaster, at $15	15.00
1 bookkeeper, at $7.50	7.50
1 cook	6.00
1 mess	22.00
Oil, lubricating	7.50
Repairs	15.00
Total	179.00

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[a] Janin, Charles, Gold dredging on the Seward Peninsula: Min. and Sci. Press, vol. 105, Sept. 28, 1912, pp. 394–399; Gold dredging in California: Min. and Sci. Press, vol. 106, Jan. 4, 1913, pp. 65–69.

A point that was not at first realized in changing the power equipment on this dredge was the decrease in weight resulting from replacing the cumbersome inefficient steam plant with electric motors. The change permitted the dredge to work without digging so deeply into bedrock, thus permitting a greatly increased yardage. The yardage handled by the dredge has been variously estimated at 2,000 to 3,000 cubic yards a day, making an operating cost for fuel used of approximately 1½ to 2 cents a yard.

POWER COST ON TWO DREDGES ON SEWARD PENINSULA.

Two dredges recently constructed on the Seward Peninsula and equipped with entirely different power arrangements are of considerable interest in showing a great reduction in power as compared to dredges having self-contained power plants using similar fuel in the northern field, and the operation of these dredges should be carefully studied by anyone contemplating dredge operation. The Berry dredge was equipped with two 75-horsepower Wolf locomobile engines using wood for fuel. The writer has obtained the following figures from the operating report of Gerald Hutton, dredge superintendent for the company; they may therefore be considered authentic.

Data covering operation of Berry dredges on the Seward Peninsula, Alaska.

[Operating days, June 3 to October 17; closed down for winter and for thawing ground.]

Net operating time (131 days—100 per cent), hours	3, 144	
Total lost time (25.7 per cent), hours	805	
Total digging time (74.3 per cent), hours	2, 339	
Total depth dug, feet	11. 9	
Total yardage handled, cubic yards	213, 355	
Total yardage handled per day, cubic yards	1, 628	
Total wood used, cords	464. 35	
Total cost of wood used, 464.35 cords, at $10	$4, 643. 50	
Cost of getting to dredge	594. 37	
Wastage, estimated	261. 89	
		$5, 499. 75
Total cost of wood per cubic yard, cents		2. 57
Total operating cost per cubic yard, cents		11. 24

The lubricants used throughout the season were as follows:

Lubricants used by Berry dredge during operating season.

Lubricant.	Total quantity used.	Per day.	Cost for season.[a]
	Gallons.	*Gallons.*	
Hecla B. cylinder oil	84	0. 64	$92. 40
Atlantic red oil	235	1. 8	141. 00
	Pounds.	*Pounds.*	
Arctic cup grease	400	3. 8	44. 00
Dixon special grease	100		16. 55
Mica gear grease	556	2¾	30. 26
Total expense for the season			324. 21
Total lubricating cost per day			2. 47
Total cost of lubricating engines per day			1. 78

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[a] Including freight and hauling. The consumption of lubricating oil on the engines will be reduced the coming season, as the oil filter was not received until fairly late in the season under consideration.

The engines gave minimum trouble, but experience gained in their use showed the advisability of having on hand certain duplicate parts in order to obviate long delay from any breakage that might occur. The time lost by the digging engine was 4 hours 20 minutes, being lost principally through inspection of the different parts. The time lost by the pump engine was 18 hours 55 minutes.

The key holding the hoisting-pump crosshead to the piston rod sheared and was replaced by a tool-steel key, which gave no further trouble. The babbitt bearing of the hoisting-pump became hot and melted, probably on account of failure of the oil feed. The brasses were replaced by spare brasses at a small expense.

Although the power cost on this dredge is high as compared to the cost of power on a dredge working in the United States and using electrical current costing from ¾ to 1½ cents for each cubic yard produced, as shown in the chapter on operating costs, the power cost per cubic yard is a remarkable showing for a self-contained dredge in the northern fields, and may reasonably be compared with the stated cost of the Three Friends dredge, as the figures given for that dredge are to be considered estimates only.

The Bangor Creek dredge operating near Nome is also equipped with 3½ cubic feet buckets. Power is supplied by two semi-Diesel Bolinder engines with a total of 140 horsepower. Oil fuel is used, a mixture of 4 parts 16° crude and 1 part 48° distillate. It is stated that the average consumption of this oil is 180 gallons a day, and the average daily yardage handled over a depth of 30 feet is 1,500 cubic yards.

The successful work of recent American dredges in the Philippine Islands has greatly stimulated the dredging industry in that field. The Guamos dredge uses wood for fuel. During 1913, according to the annual report of the operating company, the dredge was in actual operation for 5,731 hours and handled 814,500 cubic yards of material. During this period 4,500 cords of wood was consumed, costing $5 per cord delivered on the dredge. The fuel cost per cubic yard thus indicated is 1.38 cents, an encouraging figure considering the conditions under which the dredge works.

NOVEL POWER PLANT OF PHILIPPINE DREDGE.

A radical innovation was the motive power designed for the Mambuloa dredge in Ambos Camarines in the Philippines.[a] A steam turbo-generator was placed on the dredge to provide the electric current, being the first instance of such a plant being used for gold dredging. Although no reports of operation are yet available, this type of power plant has been successfully used on the United States collier *Jupiter*. The dredge plant consists of a 300-horsepower wood-burning boiler with condenser and a 625-kilowatt-ampere (500 kilowatts at 80 per cent power factor) turbo-generator running at 3,600 revolutions per minute, 440-volt, alternating-current three-phase, 60-cycle, with direct connected exciter. The main drive motor is 150 horsepower with reversible speed controller, which will reduce full speed one-third. The other motors are as follows:

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[a] Janin, Charles, New Philippine gold dredges: Mining Mag., London, vol. 13, August, 1915, p. 88.

Data on Mambuloa dredge.

	Horsepower.
Main winch	20
High-pressure pump	120
Low-pressure pump	75
Screen drive	40
Stacker	40

In addition to the main power plant there is an auxiliary steam engine of 20 horsepower which can operate the winch and move the dredge in case of accident to the turbo generator, during a storm or other emergency. This engine is used for lighting the dredge when the main plant is shut down, and can also be used to take the place of the exciter in the generator should that get out of order.

Of interest in connection with this dredge is the fact that the water for the boilers comes through a special pipe line from a fresh-water spring in the hills. This plan was adopted to obviate the possibility of damage from salt water in the river, which caused serious boiler trouble on one of the dredges in the Paracale district.

A dredge of this type is expensive in first cost, as the plant includes not only an electrically-driven dredge, but also a separate power-plant built on the dredge instead of the usual plant on shore. It is asserted by the makers that the operating cost is less under these conditions than with a separate plant on shore, as all operations are under the direct supervision of the dredge master, and only the one working crew is necessary. Should a second dredge be built, as is contemplated if results are in keeping with expectations, the power plant may be moved from the dredge and suitably rearranged on shore for two dredges. The dredge has been designed with this possibility in view, and for this reason a 625-kilovoltampere generator, which is in excess of present requirements, is placed on the dredge. Seemingly another boiler is to be added to the shore plant if that is later constructed. Although no figures of operation are yet available, details will be awaited with interest on account of the new features involved.

Suction gas engines have been tried in some dredging operations in the Federated Malay States, but such figures of actual results as are obtainable would indicate that this type of power as applied has not proved a decided success.

RECLAMATION OF LAND AFTER DREDGING IN CALIFORNIA.

HYDRAULIC MINING ENDED BY LEGISLATION.

The regulation of hydraulic mining in California in the early eighties practically abolished that form of mining in the State. California had previously been famous for its hydraulic mining, and there still remain enormous gravel deposits that could be profitably worked by hydraulic methods if such operations were permissible and if early conditions regarding water supply still remained.

In the early eighties the production of gold from hydraulic mining was about $10,000,000 a year. After injunctions and regulations had been imposed the production dropped to less than $1,000,000 a year, and in 1915 only $420,770 was obtained from this source. It can be seen that at present hydraulic mining is of relatively small importance as compared to dredge mining, which has to its credit over $7,000,000 annually since 1909 and about $8,000,000 for 1915, a total closely approximating $90,000,000 having been won by this form of mining in California from 1898 to January, 1917. According to a review of hydraulic mining by the Miners' Association in 1881, "California has produced (1848–1880) between $1,100,000,000 and $1,200,000,000 of gold, of which very much the larger portion has come either directly or indirectly from the ancient or Pliocene river channels."

At different times legislation for regulating gold dredging has been proposed, not only to prevent the accumulation of débris from the dredges, but to prohibit the dredging of land that can be used for agriculture. An act of Congress dated March 1, 1893, provides for the establishment of the California Débris Commission, to consist of three officers of the Army Engineer Corps. The main work of the commission in later years has comprised efforts to improve navigation and to control flooding of the rivers, the work relating to mining being regarded of minor importance, although the commission practically has full control of the débris situation. An excellent summary of legislation relating to mining débris is given by Charles Yale.[a]

DREDGING IN CALIFORNIA NOT INIMICAL TO AGRICULTURE.

The welfare of the agricultural interests with regard to any damages or inconveniences resulting from dredge mining has been carfully looked after by the California Anti-Dèbris Association, a protective organization first formed in the early seventies. Committees from the association have on different occasions investigated conditions in some of the dredging fields of the State, and through their efforts a number of abuses have been corrected. Stream pollution by the dredges has been considered at different times, and if the spill from a dredge was found flowing into a river remedial measures were taken. The dredge men in certain districts have for years been impounding the overflow from the dredge ponds, and have spent considerable money in preventing this water from flowing into rivers. Most of the dredges now operating in California, with the exception of those in the Yuba River, are working inland and not in the river beds proper. They are not therefore obstructing the rivers with tailing or causing overflow during flood times with resultant damage to adjoining land. A few years ago, a committee appointed by an antidredge convention reported that the dredges along the Yuba and American Rivers, instead of sending down débris, were impounding considerable sand and keeping it from going into the streams.

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[a] Yale, Charles G., Mining debris legislation: California Mines and Minerals, souvenir ed., California Miners' Association, 1899, pp. 255-262.

A bill under consideration by the California legislature in 1913 proposed a basis of limitation of the industry more severe than that of damage by pollution of streams. Its purpose was to prohibit the dredging of any land that could be used for agriculture. This agitation provoked considerable discussion, and a paper published by the writer [a] was given wide circulation at the time. A number of excerpts are taken from the paper for this chapter. Fortunately for the good of the State, as well as for the interests actively affected, the bill was defeated. A similar bill was introduced, early in 1915 but was withdrawn. It is probable that other similar bills will be introduced in the future.

SIZE OF CALIFORNIA DREDGING AREAS.

The amount of land attacked by the bill mentioned was not great. Compared to the total area of arable lands in the State, it seems almost infinitesimal. At present there is probably 12,000 acres of land suitable for dredging in the State, and only about 2,500 acres of this could be classified as agricultural. According to figures of the California Development Board, there is 28,000,000 acres of arable land in the State, and if dredging of 2,500 acres of such land were projected, it will be seen that less than 0.01 per cent of the total area of arable land would be involved. The distribution of this agricultural land is approximately as follows: Butte County, 400 acres; Yuba County, 650 acres; Sacramento County, 1,200 acres; Placer County, 100 acres; Trinity County, 50 acres; Stanislaus County, 100 acres; total, 2,500 acres.

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[a] Janin, Charles, Proposed regulation of gold dredging: Min. and Sci. Press, vol. 106, March 8, 1913, p. 381.

Nearly one-half of the total agricultural land to be dredged is owned by one company, the Natomas Consolidated of California. The loss of any small part of this land useful for agriculture is fully compensated for by the reclamation department of the company. The project will bring into production 60,000 acres of overflow land along the Sacramento River that, when fully reclaimed, will add greatly to the wealth of the State. It must also be considered that most of this overflow land was idle and practically worthless before the work of reclamation was begun, and that it would probably have been idle for a number of years to come except for the work of the Natomas Consolidated, made possible in part from the profits resulting from gold dredging. At present much of this land has been reclaimed and is being cultivated for grain, beans, rice, and alfalfa.

COMPARATIVE INCOME FROM DREDGING.

For comparative purposes the following figures may be of interest. Assuming that there is 12,000 acres of land yet to be dredged in the State, and that the average depth of this land is 11 yards, there would be an average of 53,240 cubic yards, or in round figures 50,000 cubic yards, of material to the acre. The average operating cost of dredging in the State may be taken as 5 cents a cubic yard, or $2,500 an acre. In 12,000 acres there would be distributed $30,000,000 in labor and other operating costs. The average gross value of this material may be taken as 10 cents a yard. Accordingly, $60,000,000 will be added to the production of the State from land that otherwise would have little or no productive value.

Land that was formerly planted in vineyards and handled as such at a loss has been dredged at a profit to the owner and to the State. In one area of several hundred acres the gross amount realized from grapes averaged, over a period of several years of careful cultivation and attention, about $25.40 an acre per year. The net profit was less than $10 an acre for any one year and some years the property was cultivated at a loss. The gross recovery on such of this "vineyard" land as has been dredged has averaged $2,930 an acre. Although these figures can not be applied to all vineyards, the gold produced from dredging such land may be regarded as added to the permanent wealth of the State, whereas the crops from the vineyard can hardly be regarded in any such light.

Most of the land that has been dredged in the State was originally unsuitable for either horticulture or agriculture. A considerable part of the dredging land was previously mined by methods in use prior to the development of gold dredges, and the condition of most of this land was such as to make it useless for any other purpose. Open or partly caved shafts and pits, old dumps and heaps of stones were everywhere, and practically the only vegetation was a few scrubby trees. The area of orchards and vineyards already dredged is probably not more than 2,500 acres; most of this is around Oroville. The loss resulting from destroying these orchards and vineyards, some of which had already been attacked by the phylloxers, can not be considered great. Though there are exceptional instances, it may be said that few of these orchards and vineyards returned much of a profit to their owners prior to dredging.

The removal of the tailing for use in the rock-crushing plants is a step toward the ultimate reclamation of that land. After the larger stones have been removed from the ground at Natoma, experiments have been made in planting olive and eucalyptus trees without any further treatment than the addition of a shovelful of loam around the roots of the trees as they are put in place. Although only a few acres have been planted in this way, enough has been done to demonstrate the feasibility of the plan and to give ample evidence of the fertility of this part of dredged ground. Successful experiments in planting trees and grape vines in the dredge tailing have been made at Oroville. Much of the success of the experiments was due to the abundance of clay present, which added greatly to the fertility of the tailing. The results show that the ground after dredging is not so worthless as was at first supposed. The results of these experiments have already been published elsewhere.[a]

RESOILING METHODS TRIED.

Numerous methods for resoiling behind the dredge have been suggested and some have been given thorough trial. One of these, obviously impracticable, entailed the use of a telescopic dredge ladder. Another suggestion was to use a dredge with two bucket lines, one to work on the gravel and the other to work simultaneously on the overburden. A somewhat more practical suggestion is to use an independent stripping plant to work in advance of the dredge. However, this method involves great expense in the handling of the material in front of the dredge and delivering it on the worked ground behind. The cost of reclaiming land by such a method would probably greatly exceed the value of the land.

With the Australian pump dredge [b] the material could be worked in sections and delivered advantageously to different places, but the working cost would closely approximate 25 cents a yard, an excessive cost on any ground being dredged in the United States.

Attempts to reclaim or resoil land during dredging operations have not so far been carried on to any great extent in California. Hurst [c] describes a resoiling device on a small dredge near Loomis.

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[a] Editorial. Min. and Sci. Press, vol. 105, Oct. 26, 1912, p. 541.

[b] Janin, Charles, An Australian pump dredge: Min. and Sci. Press, vol. 103, Nov. 11, 1911, p. 614.

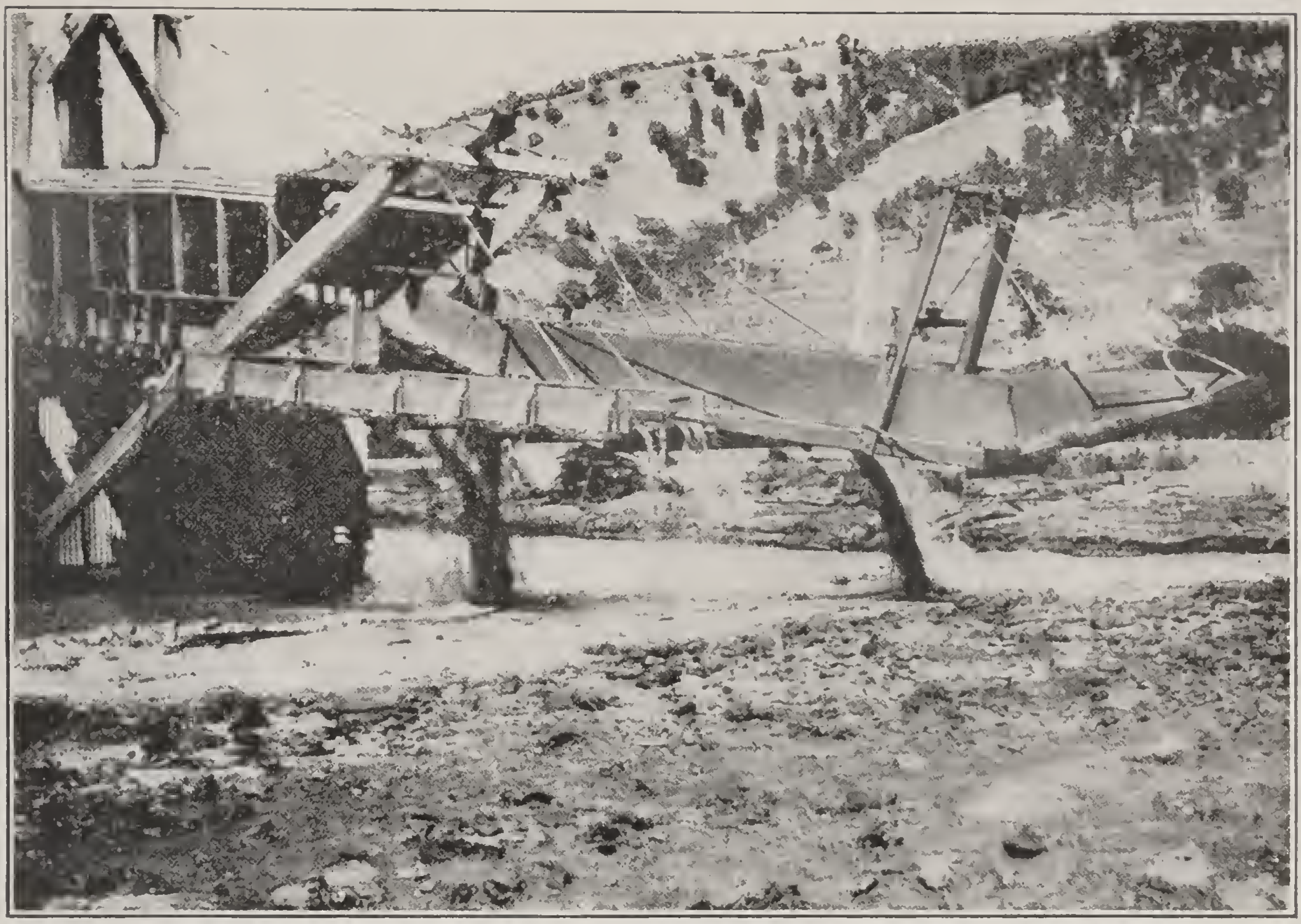
[c] Hurst, G. L., Resoiling after dredging in California: Min. and Sci. Press, vol. 107, Nov. 8, 1913, pp. 719-720.

The ground worked was only 12 to 15 feet deep and averaged about 75 per cent of fine material, about 2 feet being top soil. The principle employed was to increase the height of the upper tumbler above the deck to give greater clearance to the tail sluices. The material from the tables passed to three tail sluices, one on each side and one in the center, the center sluice passing 20 feet beyond the side sluices. The conveyor stacker was eliminated and the oversize from the screen fell on a small grizzly, the smaller material going to the central sluices and the coarser to the side sluices. The reclamation of this ground presented no unusual difficulties and a total of about 50 acres was resoiled and planted, but the device used would not prove satisfactory on much deeper and coarser gravel.

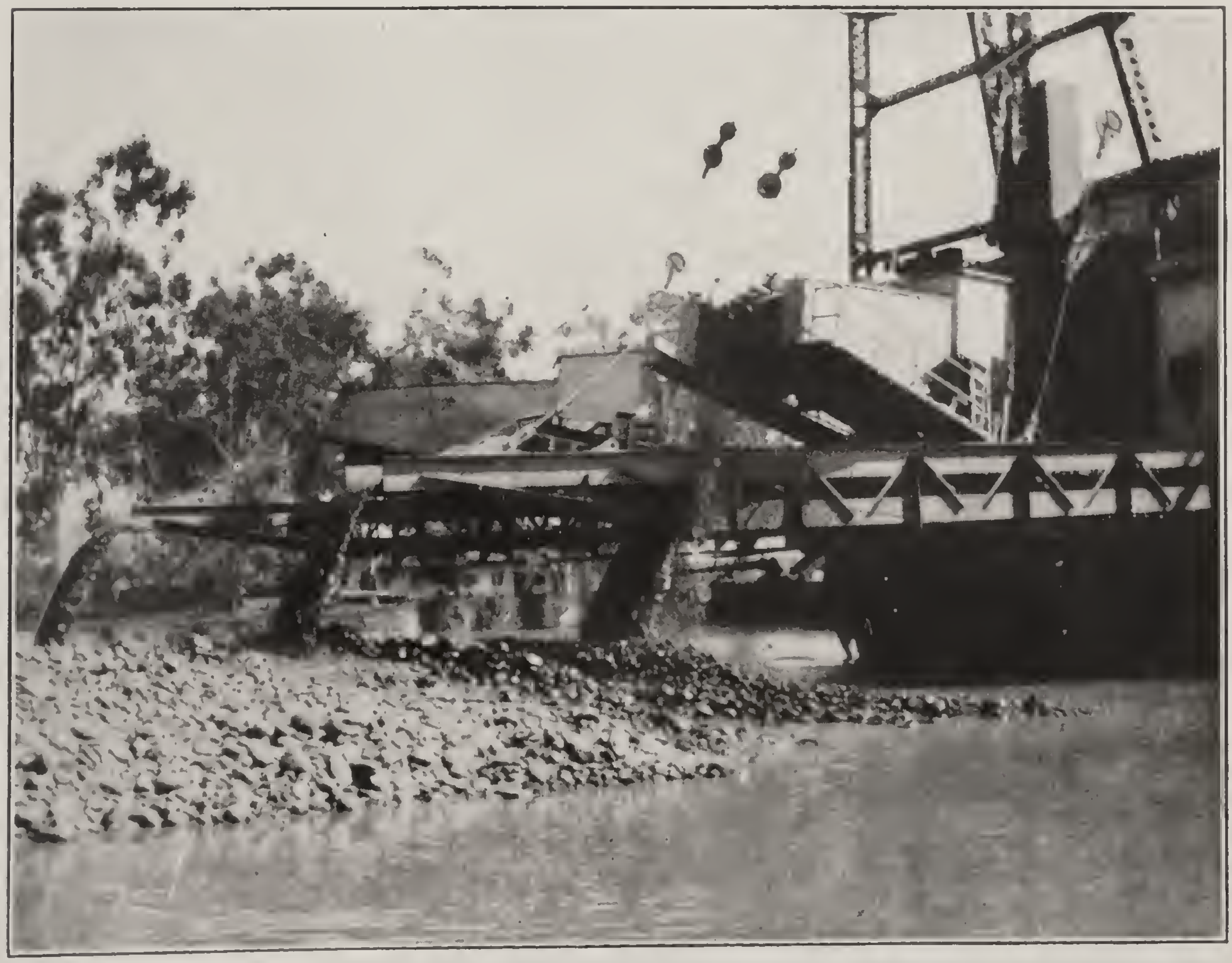
With the exceptions mentioned and a few attempts on Butte Creek in Butte County no determined effort has been made in the past to reclaim dredging land, although considerable thought has been given the matter at times by different dredging operators, and various plans for the reclamation of land already dredged has been proposed.

In addition to the gold recovered from the gravel, the reclamation of the land for agriculture might be a considerable factor in estimating the total profit to be won from the use of a mammoth dredge for this class of work. The first dredges, in turning over the ground, necessarily deposited the top soil on the bottom under the gravel and bowlders from the tailing stacker. After the greater part of the coarser gravel has been removed for use as crushed rock, with some such arrangement as that being tried in New Zealand (Pl. LXIII *A*) in resoiling experiments, this soil now below the gravel could be largely redeposited on the top of the coarser material in resoiling. Any estimate of the operating cost of a dredge of this character is, of course, pure speculation, but there seems every reason to expect that, under favorable conditions, this cost would be low.

In the Oroville field work, some encouraging recoveries have occasionally been made, when a dredge moving to a new location has redredged tailing. These in general are due to inefficient washing of the ground in the first instance, or to the dredge digging in a part of the bedrock section that had been passed over in the first dredging. However few dredge operators think that this land can be redredged at a profit, although a much lower operating cost has been shown by the dredges when working through tailing. Unquestionably this material can be handled much easier than ground that has never been dredged, especially in certain sections of Oroville where the gravel is cemented.



A. SLUICE BOX AND SOIL CHUTE OF HINNOMUNJE DREDGE, OMECO, NEW ZEALAND.



B. TAILING DISTRIBUTOR ON NATOMA NO. 4 DREDGE.

LAND RECLAMATION DEMANDS DREDGE CHANGES.

To reclaim land while the dredge is at work it is necessary to make some radical changes in the distributing end of the dredge, which as has been stated, is not difficult when shallow land is to be dredged. In land with a deposit of 20 feet or more, the problem is not so easy. Considerable thought has been given to this matter by the engineers of the Natomas company and a dredge has recently been built embodying their ideas to solve the problem of replacing the soil behind the dredge in suitable condition for farming. In order to get sufficient clearance for the tail sluices all the machinery is raised about 6 feet higher than usual. To obtain a more even distribution of the coarse material two tailing stackers are fed from a double hopper arranged at the end of the revolving screen to divide equally the material supplied each. The dredge is equipped with double-deck tables and tail sluices which are set much closer than usual, a number of sluices on the lower tables being omitted in order to allow room for cleaning up, etc. The double tailing-stackers which are shorter than usual, being 56 feet long with an elevation of 1 inch in 12 inches as compared to the usual stacker inclination of 18° (or 3.7 inches in 12 inches) are mounted on swivels, as are also the side sluices. It is not expected to swing these tailing distributors from side operation, as the swing of the dredge will provide for lateral distribution. The swivel arrangement is used only to insure horizontal adjustment for these distributors under different conditions of operation. Plate LXIII, *B*, shows the tailing distribution of this dredge at work.

The dredge started work in January, 1916, and operations are being watched with interest. A number of changes have been made since the dredge first started work when it was found the arrangements originally planned were not successful, and after the problem of recoiling is effectively solved similar arrangements will be adopted on No. 1 dredge when that boat is rebuilt. The land that is to be dredged is 20 to 40 feet deep and a great part of it is planted to vineyard. If an extra cost of one-half cent a yard is caused by experimentation, special equipment, and operation, and assuming that the ground to be dredged will average 40,000 yards to the acre, an extra cost of $200 an acre will be incurred. After the problem of resoiling coincidental with dredging has been solved, the extra cost involved will be small but at present the Natomas company, working for the best interests of the State, is going to considerable more expense than could be borne by a smaller company.

FUTURE OF DREDGING.

The highest point of production for the gold-dredging industry in the United States has been reached, and during the next few years a marked falling off in gold production from dredging operations may be expected. The success attending the flotation of the larger dredging companies proved an incentive for explorations and examinations in various States, and all land thought attractive for dredging was soon taken. The problem of finding suitable ground for dredging is difficult, although with the great improvements made in dredging machinery and the resultant reduction in operating costs, it is obvious that gravel beyond the possibility of economic dredging under early conditions could now be profitably handled. This is shown by the operations of nearly all the large dredging companies whose available dredging ground has been greatly increased as working costs have been reduced.

A fact that also draws attention to areas once considered unprofitable is that the life of some of the successful dredging companies is drawing to a close; that is, the land for the working of which the companies were originally formed is being rapidly worked out. Several of these companies are realizing that their ground is nearly exhausted and that they have some dredges in good condition, fit for some years of operation. What is probably more important, they also realize that they have also well-organized working forces, valuable experience in dredging matters, money in the treasury, and the confidence of shareholders.

Aside from such influences, there is no question that under favorable conditions areas that were once considered unsuitable for economic dredging can now, with improved methods, be profitably handled. The possibilities of reclamation of dredged land may prove important in estimating the profits to be won from dredging areas in California. The success attending the efforts of some experiments in this direction in a small way has greatly broadened the expectations for the future of what was, until recently, considered worthless land. A more liberal attitude on the part of the agriculturists and arrangements permitting the modification of some of the rulings of the Debris Commission in California would open to the dredging industry areas that are at present closed.

FEASIBILITY OF REDREDGING CERTAIN AREAS.

The possibility of redredging some of the tailings from the early dredges at Oroville, partly with the idea of reclaiming lands that would command a fair market, is occasionally discussed in the nontechnical press. This proposal has not met with any enthusiasm among the conservative operators, but it should be realized that the greater improvements in the dredging industry are not due to the most conservative men; one must be optimistic to undertake new methods. The reclamation of ground after dredging is discussed in a previous chapter. Altogether it seems probable that dredging enterprises will continue in California for some years to come, although the amount of gold won from dredging operations in the State was at its peak in 1913. During 1915–16 there have been a number of new dredges planned and at the time this is written there are nine new dredges either under construction or projected for the California fields alone. Four of these are 16-foot boats, three 9-foot, and two are old dredges being moved to new locations. The writer pointed out in a paper read before the San Francisco meeting of the American Institute of Mining Engineers in California, 1911,[a] that dredges of even larger capacity than those now in use for gold dredges had been considered in the reclamation of dredged areas. The following extract from the paper is pertinent:

> With the development of the gold dredge to its present efficiency, the question is often raised as to when the limit in size for economic dredge installation will be reached. Much depends upon the conditions met in operation. There is no question as to the mechanical possibility of larger buckets. In Boston harbor a bucket-elevator dredge equipped with buckets of 2 cubic-yard capacity has been successfully operating for some years on harbor work, and on the Danube River in Germany a bucket-elevator dredge having 2.5 cubic-yard buckets is now in operation. While the mechanical possibilities have thus been proved, to apply such radical changes in size to the gold dredge of to-day would necessitate an entirely different arrangement of the gold-saving tables and would probably result in a general modification of the whole gravel-washing apparatus now in use. Even the most optimistic advocates for increasing the size of the dredge buckets would hesitate at recommending a 2 cubic yard bucket, which is nearly four times the present size of the buckets on the largest gold dredges in operation, but there are a number of engineers who believe that the bucket-elevator dredge with buckets having a capacity of 1 cubic yard will be constructed before long. While a dredge of this character would necessarily be equipped with heavier machinery and a larger hull than those on the present 15 cubic-foot boats, it is, as before stated, quite possible that, with modifications of the washing apparatus, the hull of the 1 cubic-foot dredge may not be proportionately larger. The present 15 cubic-foot boats have a hull 60 by 150 by 12.5 feet, with a deck overhang of 5 feet on either side, making a total width of 70 feet. The gold-saving tables are of the double-bank type and have an approximate area of 7,000 to 8,000 square feet. Without some change in the washing apparatus, it can readily be seen that 14,000 square feet of table area would necessitate a hull of greatly increased size, or additional tiers of tables, for which an increased length of bucket ladder would be required to elevate the gravel to the additional height, or a general change in the design of the boat. Practice has demonstrated that when digging free-washing gravel the table area of the 15-foot boats is considerably in excess of all requirements, and some operators contend that it would not be necessary to proportionally increase the table area when buckets of 1 cubic-yard capacity are constructed.

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[a] Janin, Charles, Present-day problems in California gold-dredging: Trans. Am. Inst. Min. Engrs., vol. 42, 1912, pp. 855–873.

There may be a field for dredges of this size, for instance, in the Oroville and Folsom fields, to redredge the tailing piles left from the first dredging operations. After many of the cobbles have been removed for the rock-crushing plants, the ground, if dredged, will, in many cases, yield a fair return. Especially would this be the case in the areas where the early dredges worked, as the gold-saving apparatus of the first successful dredges was not as efficient as that in present use.

ALASKA FIELDS PROMISING.

In Alaska the success attending the operations of some of the dredges recently built there has greatly widened the field for the dredge-construction companies. Previously the demand has been for small dredges, especially on the Seward peninsula, but the successful operation of the large dredges of the Canadian Klondike Co. on the Yukon, and the Bonanza Creek operations of the Yukon Gold Co. have encouraged investigation of areas in other parts of the northern field. Unfortunately, many of the dredges built in Alaska have been constructed prior to adequate prospecting of the ground they were to work. Failure to sample a dredging property until the dredge has been built, when "large practical samples" can be taken, is almost to invite loss. Although the dredge may prove a success mechanically, the company is generally doomed to economic disaster unless more favored with good luck than good judgment.

Besides the possibility of new dredging enterprises in California and Alaska, the attention of dredge operators is being attracted to areas in other western States where the gold content has previously been considered too low. The difficulties surmounted in dredging operations in California, Montana, Colorado, and other fields, and the low working costs obtained in these fields and in Idaho, open possibilities unthought of a few years ago.

DREDGING OUTLOOK IN OTHER COUNTRIES.

In addition to the possibilities in the United States and Alaska, considerable attention has been given to foreign fields. Chief among the countries being considered are Russia, including Siberia; South America, principally meaning Colombia; and the federated Malay states. The success attending the work of some Russian dredges and the purchase early in 1916 of three American dredges for the Russian field has stimulated inquiry into the possible openings for American capital in dredging projects in that field. It is rumored that the Russian Government has considered the advisability not only of taking off the high duty heretofore imposed on machinery for gold mining, but also of subsidizing to some extent gold-mining companies; such procedure would undoubtedly stimulate investigations of dredging areas in that country. In new enterprises in Russia or Siberia, it will be found advisable to have the assistance of influential Russians. It is held by engineers qualified to pass an opinion that in Russia and Siberia, so far as operating conditions are concerned, there are a number of areas of gravel that can be profitably dredged, and the examination of some of these fields may result in more dredges being erected in the near future. A pertinent point in this connection is that the Russian peasant in some localities makes a really first-class mechanic, so that it is not necessary for a foreign dredging company to import expensive laborers for all positions on a dredge. Conditions of operation in Russia and Siberia, aside from taxes and government regulations, are very similar to those in the Yukon and Alaska, and operating costs should, under ordinary conditions, be closely similar. The success made by the gas, oil, and wood engines in Alaska, described elsewhere, should encourage dredge installations in localities where electric power is not obtainable.

There are undoubtedly in Colombia and other South American countries areas that offer promising returns from dredging operations. A number of investigations have been made in some of the countries in the interest of American capital, but the difficulties of transportation, of climate, and of procuring sufficient cheap power have proved somewhat discouraging. Here also some of the improvements in power arrangements for the self-contained dredge will be worth consideration.

In the Philippines the success of the Guamos dredge has encouraged other operations and an American dredge built in 1914 for the Mambulao Dredging Co. embodies some striking innovations for generating power on board the dredge. This dredge is described elsewhere (see p. 196) the results of its operation are awaited with interest.

An American dredge was constructed for the Chiksan Mining Co. in Korea during the latter part of 1916. Its success will do much to encourage further dredge installation on the Orient. In Portugal a dredge built under plans of an American dredge-construction company for American capitalists is being operated in tin deposits, and its success has awakened interest in tin dredging. In 1915–16, several orders for new dredges for the Malay field were placed with American dredge-construction companies, and this field offers considerable promise to the wide-awake constructor.

Although there is room for considerable improvement in the gold dredge, the wasteful and inefficient methods usually employed on tin dredges should give our dredge-construction engineers a splendid opportunity for increasing the recovery. The profits made by some of the tin dredges have been large, and doubtless no necessity has yet been felt for inquiring closely into methods for decreasing the loss in the dredge tailing. It is when low-grade material is handled that metallurgical methods are improved. It is probable that American operators will be offered more or less attractive opportunities to participate in some of these tin-dredging projects in the near future. Altogether the possibilities of successful dredging operations in some of the foreign countries mentioned seem to be attractive.

SUCCESSFUL DREDGING OPERATIONS DEMAND COMPETENT EXPERTS.

In making investments in dredging enterprises in foreign countries, as elsewhere, much money may be saved if prospective investors will insist on having reports by experienced engineers. With the drawbacks of operating in foreign countries, together with unusual government regulations, and other disadvantages incident to dredge work away from home, possible dredging areas in the United States, though of lower grade than some that are being at present worked, and of much lower gold content than is necessary for profitable operation in foreign countries, will probably be more attractive to conservative investors than the foreign fields. Many investors, however, will be attracted by the higher gold content claimed for the gravels in foreign fields, and some operators, acting on the reports of competent engineers, will risk enterprises in foreign countries, and, profiting by their knowledge of the industry, will build suitable dredges and will realize large returns on the initial investments.

Close estimates of the possible profits from gold-dredging projects can be made, but competent experts must have charge of the details of examination and prospecting and the development operations must be under the direction of efficient management, which entails competent engineering advice.

LIST OF DREDGING OPERATIONS.

A fairly complete list of dredging operations throughout the world follows. The list does not include Australian, New Zealand, nor most Siberian dredges. In a table of this kind full credit cannot always be given the makers, as, for instance, reconstructed dredges and dredges that were constructed by one company and part of the machinery is furnished by another.

Incomplete list of all dredging operations.[a]

[Not inclusive of Australian, New Zealand, nor most Siberian dredges.]

Name of company.	Headquarters.	Situation of dredge.	Make of dredge.	Bucket capacity.
				Cu. feet.
Alaska Gold Dredging Co.	Council, Alaska	Council, Alaska	Special	2½
Alaska-Kougarok Dredging Co.	Nome, Alaska	Kougarok River	do	2¾
Alta Bert Gold Dredging Co.	Trinity Center, Cal.	Trinity Center, Cal.	Union Con	7½
American Gold Dredging Co.	Nome, Alaska	York, Alaska	American	1½
Do	do	do	do	1½
American Placer Mining Co.	Spokane, Wash	Pierce, Idaho	Risdon	3½
American Tin Co.	San Francisco, Cal.	York, Alaska		
Amur Gold Mining Co.	Blagovestensk, Siberia.	Kharga River	Conrad	5
Anderson, Andy	Nome, Alaska	Center Creek	Union	3½
Andrada Mines (Ltd.)	London	Macequece, Portuguese East Africa.	Bucyrus	7½
Anglo Alaskan Gold Dredging Co.	Nome, Alaska	Nome, Alaska	Special	2
Arctic Gold Dredging Co.	do	Hobson Creek	Union Con.	2¾
Bangor Dredging Co.	do	Bangor Creek	do	3½
Behring Dredging Co.	do	Teller Creek	do	2½
Berry Dredging Co., C. J.	Circle, Alaska	Mammoth Creek	do	3½
Blue Goose Mining Co.	Council, Alaska	Ophir, Alaska	Reconstructed	3½
Bohannon Dredging Co.	Salmon, Idaho	Salmon, Idaho	Special	5
Borden Co., Ivy L.	San Francisco, Cal.	Jenny Lind, Cal.	Bucyrus	6
Boston & Idaho Gold Dredging Co.	Idaho City, Idaho	Idaho City, Idaho	Yuba	5
Bourbon Dredge	Nome, Alaska	(See Nome Holding Co.)		
Budd Creek Gold Dredging Co.	do	Budd Creek	Special	3
Butte Dredging Co.	San Francisco, Cal.	Jenny Lind, Cal.	do	3½
Canadian Klondike Mining Co. (Ltd.).	Dawson, Alaska	Dawson, Alaska	Marion	7½
Do	do	do	do	16
Do	do	do	do	16
Do	do	do	do	16
Candle Creek Mining Co.	Candle, Alaska	Candle Creek	Union	1¾
Casadepaga Gold Mining & Dredging Co.	Solomon, Alaska	Casadepaga River	Special	2½
Cie. des Mines de la Faleme	Paris, France	Senegambia	Lobnitz	7
Chestate Dredging Co.	Gainesville, Ga	Gainesville, Ga	Bucyrus	3½
Chiksan Mining Co.	Chiksan, Korea	Chiksan, Korea	New York Engineering.	10
Compagnie Francaise du Mataroni.	Cayenne, France	French Guiana	Lobnitz	3½
Conrey Placer Mining Co.	Ruby, Mont	Ruby, Mont	Marion	7½
Do	do	do	do	7½
Do	do	do	do	9
Do	do	do	do	16
Deering Dredging & Mining Co.	Deering, Alaska	Deering, Alaska	Special	2½
Derry Ranch Gold Dredging Co	Leadville, Colo	Malta, Colo	New York Engineering.	6
El Oro Dredging Co.	do	Yreka, Cal.	Special	5
Empire Gold Dredging Co.	John Day, Oreg	John Day	Stearn Rodgers.	5
Ernst Alaska Gold Dredging Co.	Nome, Alaska	Nome, Alaska	Special	1¾
Federal Mines Co.	Fitting, Nev	Unionville, Nev.	do	5½
Flodin Gold Mining & Dredging Co.	Chicago, Ill	Solomon, Alaska	do	2½
Flume Dredge Co.	Nome, Alaska	Ophir Creek	Straub	2½
Do	do	Crooked Creek	do	2½
Freeze Dredging Co.	do	Nome, Alaska	Special	2
French Gulch Dredging Co.	Breckenridge, Cal.	Breckenridge, Cal.	Bucyrus	5

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[a] Based on list from Eng. and Min. World, Jan. 6, 1916.

Incomplete list of dredging operations—Continued.

Name of company.	Headquarters.	Situation of dredge.	Make of dredge.	Bucket capacity.
				Cu. feet.
Gardella, Lawrence	Oroville, Cal	Honcut, Cal		5
Do	do	Garvin, near Redding, Cal.		5
Gumaos Placer Co	Manila, P. I	Guamos, P. I	New York Engineering.	6
Hastings Creek Dredging Co	Nome, Alaska	Hastings Creek	Special	1½
Holyoke Gold Dredging Co	do	Nome, Alaska	do	9
Idaho Power Co	Spokane, Wash	Pierce, Idaho	do	3½
Immachuck Dredging Co	Deering, Alaska	Immachuck River	Union Con	3
Isabel Gold Dredging Co	(See Borden Co., Ivy L.)			
Johnson, Iver	Candle, Alaska	Kugruk River	Union Con	3½
Julien Gold Mining & Dredging Co.	Nome, Alaska	Osborne Creek	do	2¾
Kelliher Dredging Co	do	Nome, Alaska	Special	2¾
Kentucky Ranch Gold Dredging Co.	Oroville, Cal	(See Gardella, Lawrence.)		
Kimball Dredging Co	Solomon, Alaska	Adams Creek	Special	2½
Kirtley Creek Gold Dredging Co.	Salmon, Idaho	Salmon, Idaho	Yuba	9
Kugruk Dredge (see Iver Johnson).	Candle, Alaska	Kugruk River	Union Con	3½
La Grange Gold Dredging Co	La Grange, Cal	La Grange, Cal	Special	7
Lincoln Dredge	Auburn, Cal	(See Gardella, Lawrence.)		
Lubbe Gold Dredging Co	Council, Alaska	Council, Alaska	Special	2½
Malaguit Dredging Co	Manila, P. I	Parcale, P. I	Yuba	5
Malayan Tin Dredging (Ltd.)	London, England	Btu Gajah, Perak	Lobnitz	3¾
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	5
Mambuloa Placer Co	Manila, P. I	Mambuloa, P. I	New York Engineering.	8
Marysville Dredging Co	Marysville, Cal	Marysville, Cal	Marion	8
Do	do	do	do	8
Do	do	do	Union Iron W'ks	16
Merced Dredging & Mining Co	Snelling, Cal	do	Yuba	3¾
Natomas Consolidated of California (Feather River).	San Francisco, Cal	Oroville, Cal	Yuba	8
Do	do	do	do	8
Do	do	do	do	15
Natomas Consolidated of California.	do	Natoma, Cal	do	13½
Do	do	do	Bucyrus	8½
Do	do	do	do	8½
Do	do	do	do	9
Do	do	do	do	9
Do	do	do	do	9
Do	do	do	do	13½
Do	do	do	Yuba	15
Do	do	do	do	15
Do	do	do	Bucyrus	15
Nechi Mines, Colombia	(See Oroville Dredging, Ltd.)		Union and Fraser & Chalmers.	8½
Nicola Pavda Mining District Co.	Petrograd, Russia	Perm	Marion	7½
Nome Holding Co.[a]	Seattle, Wash	Nome, Alaska	Special	10
Do	do	do	do	7
Do	do	do	do	7
Northern Lights Mining Co	Council, Alaska	Ophir Creek	Union Iron W'ks	3
Oroville Dredging (Ltd.)	San Francisco, Cal	Oroville, Cal	Yuba	7
Do	do	do	do	7
Do	London, England	Nechi mines, Zaragosa, Colombia.	Union and Fraser & Chalmers.	8½
Do	do	Pato mines, Zaragosa, Colombia.	Yuba	9
Oroville Union Gold Dredging Co.	San Francisco, Cal	Oroville	Special	6½
Oro Water, Light & Power	do	do	New York machine.	6
Do	do	do	do	5
Do	do	Camanche, near Clements, Cal.	do	9
Do	do	do	do	6
Orsk Gold Fields	London, England	Nikolaievsk, Siberia.	Yuba and New York Engineering.	5
Do	do	do	New York Eng.	8

[a] Formerly Nome Consolidated Dredging Co.

Incomplete list of dredging operations—Continued.

Name of company.	Headquarters.	Situation of dredge.	Make of dredge.	Bucket capacity.
				Cu. feet.
Otter Dredging Co	Iditarod, Alaska	Otter Creek	Union Con	3½
Pacific Gold Dredging Co	New York, N. Y	Oroville	Bucyrus	7½
Do	do	Chico, Cal	do	5
Do	do	Auburn, Cal	do	7½
Do	do	Carville, Cal	Marion	9½
Parcale Bucket Dredging Property (Ltd.)	Parcale, P. I	(See Philippine Dredges, Ltd.)		
Pasadena Dredging Co	Teller, Alaska	Teller, Alaska	Special	3
Pato Mines	(See Oroville Dredging Ltd.)			
Pavda, Nicola, Mining District Co.	Petrograd, Russia.	Perm	Marion	7½
Philippine Dredges (Ltd.)	Parcale, P. I	Parcale	Australian	5
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	7
Portuguese American Tin Co	Lisbon, Portugal	Portugal	Union Con	4
Powder River Gold Dredging Co.	San Francisco, Cal	Sumpter, Oreg	Yuba	7
Do	do	do	do	9
Plein Mining & Dredging Co	Nome, Alaska	Nome, Alaska	Special	3¾
Riverton Dredging Co	Shoshoni, Wyo	Shoshoni, Wyo	No recent report.	
Ropp Tin (Ltd.)	London, England	Bukeru, No. Nigeria.	Bucyrus	2½
Do	do	do	do	2½
Ruby Gold Dredging Co	Solomon, Alaska	Casadepaga River	Union	2¾
Santa Fe Dredging Co	Golden, N. Mex	Golden, N. Mex	Special	
Saunders Creek Dredging Co	Nome, Alaska	Saunders Creek	do	3½
Saupe Placer Mining Co	Council, Alaska	Council, Alaska	do	2¾
Seward Dredging Co	New York, N. Y	Nome, Alaska	Bucyrus	3½
Shasta Dredging Co	San Francisco, Cal	Larkin, Cal	Yuba	5
Sievertson Johnsen Mining & Dredging Co.	Solomon, Alaska	Solomon, Alaska	Special	2½
Sioux Alaska Mining Co	Nome, Alaska	Hastings Creek	Union	2¾
Siskiyou Dredging Co	Hammonton, Cal	Fort Jones	Yuba	5½
Tin Bentong (Ltd.)	London, England	Kuala Lumpur, F. M. S.	Bucyrus	7
Tincup Gold Dredging Co	Denver, Colo	Tin Cup, Colo	Marion	3½
Tin Mines of Siam (Ltd.)	London, England	Pong District, Siam	Bucyrus	5½
Tonopah Placers Co	Breckenridge, Colo	Breckenridge, Colo	do	9½
Do	do	do	do	9½
Do	do	do	do	9
Trinity Gold Dredging Co	Lewiston, Cal	Lewiston, Cal	Special	11
Uplift Mining Co	Council, Alaska	Council, Alaska	Union	2
Valdez Dredging Co	San Francisco, Cal	Junction City, Cal	Union Iron W'ks	7
Warm Creek Dredging Co	do	do	Special	2½
Welch, J. M	Teller, Alaska	Windy Creek	Union	3
Wild Goose Mining & Trading Co.	Ophir, Alaska	Golovin Bay	Yuba	3½
Willow Creek Dredging Co	San Francisco, Ca	Council, Alaska	Special	2½
Wonder Dredging Co	Nome, Alaska	Nome, Alaska	(See Nome Holding Co.)	
York Dredging Co	York, Alaska	Buck Creek	Union	2½
Yosemite Dredging & Mining Co.	Snelling, Cal	Merced Falls	Yuba	3¾
Yuba Consolidated Gold Fields.	Hammonton, Cal	Hammonton, Cal	Bucyrus and Yuba.	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	15
Do	do	do	Yuba	15
Do	do	do	do	16
Yukon Gold Co	New York, N. Y	Ruby, Alaska	Union Con	3½
Do	do	Iditarod, Alaska	Special	7½
Do	do	Dawson, Alaska	Marion	7
Do	do	do	Special	7
Do	do	do	Bucyrus	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	7
Do	do	do	do	5
Do	do	do	do	5
Do	do	do	do	5

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Bulletin 128 Mineral Technology 18

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

REFINING AND UTILIZATION OF GEORGIA KAOLINS

BY

IRA E. SPROAT

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PREFACE.

For years American manufacturers of tile and white ware have been dependent on the importation of foreign kaolin, chiefly in the form of English china clay, for their high-grade material. In 1916, for example, 229,093 tons of such kaolin were imported, valued at $1,377,106, at the port from which it was shipped. At the same time there were known to be available in Georgia and South Carolina large quantities of kaolin of a high degree of purity, except that it contained certain color-giving minerals, especially iron and titanium compounds, and had an abnormal shrinkage. Heretofore this material has been available only for the paper trade and for some lower grades of pottery. Investigations carried on by the division of mineral technology of the Bureau of Mines have shown that the impurities may be readily and cheaply separated and the kaolin so modified in its physical characteristics, both as to shrinkage and to the development of color on burning, that it is comparable with the best English fire clay for many pottery purposes.

By the addition of a small amount of sodium hydroxide, this amount being carefully controlled by chemical analysis, the kaolin colloids may be so deflocculated that the clay remains in suspension in water through long periods of time, allowing the small particles of heavier minerals to settle out readily. On neutralizing afterwards with sulphuric acid, the clay readily settles and may then be filtered and dried as usual.

As a result of laboratory experiments, cooperative experiments were conducted with the Georgia Kaolin Co. at Dry Branch, Ga., and a plant built in accordance with the facts discovered in the laboratory. As a result, many tons of kaolin have been purified at a cost of less than 50 cents a ton, producing thereby a material which has a market value nearly twice that which could be obtained for the kaolin for the purposes to which it was previously put. The purified china clay produced has been further investigated in potteries on an extensive scale through the cooperation of the Zanesville Mosaic Tile Co., the Mayer China Co., and the Beaver Falls Art Tile Co. It has been shown that in vitreous chinaware this purified American kaolin can be successfully substituted for all of the ball clay heretofore used and for at least 50 per cent of the English china clay. Better still, in the tile industry whiter and stronger tile can be and are being made from this treated Georgia kaolin and American feldspar by substituting them entirely for the English china clay and Cornwall stone heretofore used.

America should, accordingly, be largely independent, and, with further experiments, probably entirely independent of imports of foreign material for the white-ware industries. The details of the work are described in the following pages.

CHARLES L. PARSONS,
Chief, Division of Mineral Technology.

REFINING AND UTILIZATION OF GEORGIA KAOLINS.

By Ira E. Sproat.

INTRODUCTION.

The question "Why can not American kaolins be substituted for English china clay?" has been asked time and time again, and more often since the beginning of the present war in Europe, which has threatened to cut off the supply of European clays imported from abroad. The answer to this question has invariably been, that the domestic kaolins are not as pure and uniform in composition as the imported clays.

This demand for white-burning American clays is of such economic importance that it was considered advisable to determine whether some of our vast deposits of impure white clays could not be refined sufficiently to permit their being substituted for the foreign materials.

Up to the present time mechanical principles only have been applied in the refining of kaolins, but in order to keep pace with the increasing requirements for better quality and uniformity of product, the application of the principles of colloidal chemistry is necessary.

The investigation described in this report was carried on to determine the practicability of applying technical control of clay disperse systems to the refining of kaolins and the utilization of the prepared clay in the manufacture of vitreous china and wall tile.

It is hoped that this report will prove a stimulus to the clay-washing industry; will point out a method of refining certain American clays, and will lead to the substitution of domestic kaolins for imported china clays by manufacturers of high-grade ceramic wares.

SECONDARY KAOLINS OF GEORGIA AND SOUTH CAROLINA.

Immense deposits of secondary kaolins underlie certain areas in Bibb, Twiggs, Wilkinson, Washington, Glasscock, Jefferson, and Richmond Counties, in Georgia; and Aiken, Edgefield, Lexington, and Kershaw Counties, in South Carolina. As these sedimentary kaolins in the Cretaceous strata of Georgia and South Carolina occur in large deposits which run very high in clay substance, and in some of their physical properties are not unlike the English china clay, it was thought advisable to select these kaolins as a basis for the experiments.

PRINCIPAL DEPOSITS OF SECONDARY KAOLINS.

A general study of the principal kaolin deposits of South Carolina and Georgia was made by Robert Back, formerly of the Bureau of Mines, in 1913, under the direction of C. L. Parsons, chief mineral technologist. The following description of the nature of the different kaolin deposits of Georgia and South Carolina is taken from Mr. Back's field notes, supplemented by the writer's observations in the Dry Branch district of Georgia.

There are two great areas; the Aiken district, in Aiken County, South Carolina, and the Dry Branch district, in Bibb and Twiggs Counties, Georgia, which are being worked extensively to-day. There are a number of other deposits both in Georgia and South Carolina which in quality and extent rival those of the Dry Branch and Aiken areas, but they are located farther from railroads and have not been worked to any great extent. The deposits in Georgia have been described in detail by Veatch,[a] and those of South Carolina by Sloan.[b]

KAOLIN MINING IN THE AIKEN DISTRICT, SOUTH CAROLINA

Kaolin is mined in the Aiken district from open cuts with gravity drainage or from pits which are drained by steam pumps. A number of kaolin mines have been opened in this district, chief of which are those of McNamee & Co., South Carolina Clay Co., Paragon Kaolin Co., Immaculate Kaolin Co., and Peerless Clay Co.

MCNAMEE & CO.

The property of McNamee & Co. is situated about one mile from Bath and near Horse Creek. The kaolin is mined by open cuts having gravity drainage. The overburden, which is about 30 feet in depth, is removed with a steam shovel. The clay bed is known to attain in places a thickness of about 15 feet, the upper 2 feet and lower 7 feet of which are full of grit and sand and are thrown away. This clay is simply air-dried and sold to the paper trade.

SOUTH CAROLINA CLAY CO.

The South Carolina Clay Co. operates an open-cut mine near Horse Creek and not far from that of McNamee & Co. The kaolin deposit is about 8 feet thick and the overburden ranges from 20 to

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[a] Veatch, Otto, Second report on clay deposits of Georgia: Bull. 18, State Geol. Survey, 1909, 453 pp.

[b] Sloan, Earle, A preliminary report on the clays of South Carolina: Bull. 1, ser. 4, State Geol. Survey, 1904, 171 pp.

40 feet thick. The top 8 to 10 inches of kaolin, which is badly stained with iron, is thrown away, and the underlying kaolin, yellowish in color, is dried in the open air and sold to the pigment and crayon trades.

PARAGON KAOLIN CO.

The mines of the Paragon Kaolin Co. are about 1½ miles from Langley Station. This company is operating two open-cut mines with a daily capacity of about 100 tons. The clay bed is about 12 feet thick. The overburden, which varies in thickness from 30 to 40 feet, is removed with mules and carts. The kaolin obtained from these two mines is rather soft and mealy, and the output is sold exclusively to the paper trade.

IMMACULATE KAOLIN CO.

The mine of the Immaculate Kaolin Co. is located about 2 miles southeast of Langley Station. The clay deposit in this mine reaches a thickness of about 25 feet and is very white and free from grit, especially in the lower part of the bed. The overburden, which reaches a thickness of 40 feet, is removed with a cable excavator. The clay is sold directly and exclusively to the paper trade.

PEERLESS CLAY CO.

The Peerless Clay Co. operates a mine 2 miles southeast of Langley Station, the clay deposit being very similar to that mined by the Immaculate Kaolin Co.

KAOLIN MINING IN THE DRY BRANCH DISTRICT, GEORGIA.

In March, 1916, there were three companies actively engaged in the mining and marketing of secondary kaolins within a radius of three miles of Dry Branch. The mines are all open-cut mines.

GEORGIA KAOLIN CO.

The Georgia Kaolin Co. is the only one in this district actively engaged in washing kaolin. The clay as mined is highly plastic, has a high strength, and burns to a good white, but if glazed the color becomes a decided cream. The product is sold to the paper trade, pigment manufacturers, and some ceramic industries. A view of a face in this mine is shown in Plate I, *A*.

AMERICAN CLAY CO.

The American Clay Co. sells its entire output to the paper trade. The clay is not washed and the only mechanical treatment it receives is crushing after being air-dried. A washing and refining plant is being built. This clay has a good plasticity and burns to about the same color as the clay mined by the Georgia company, but is softer and more mealy and has a lower tensile strength. A view of the claybank in this mine is shown in Plate I, *B*.

JOHN SANT CLAY CO.

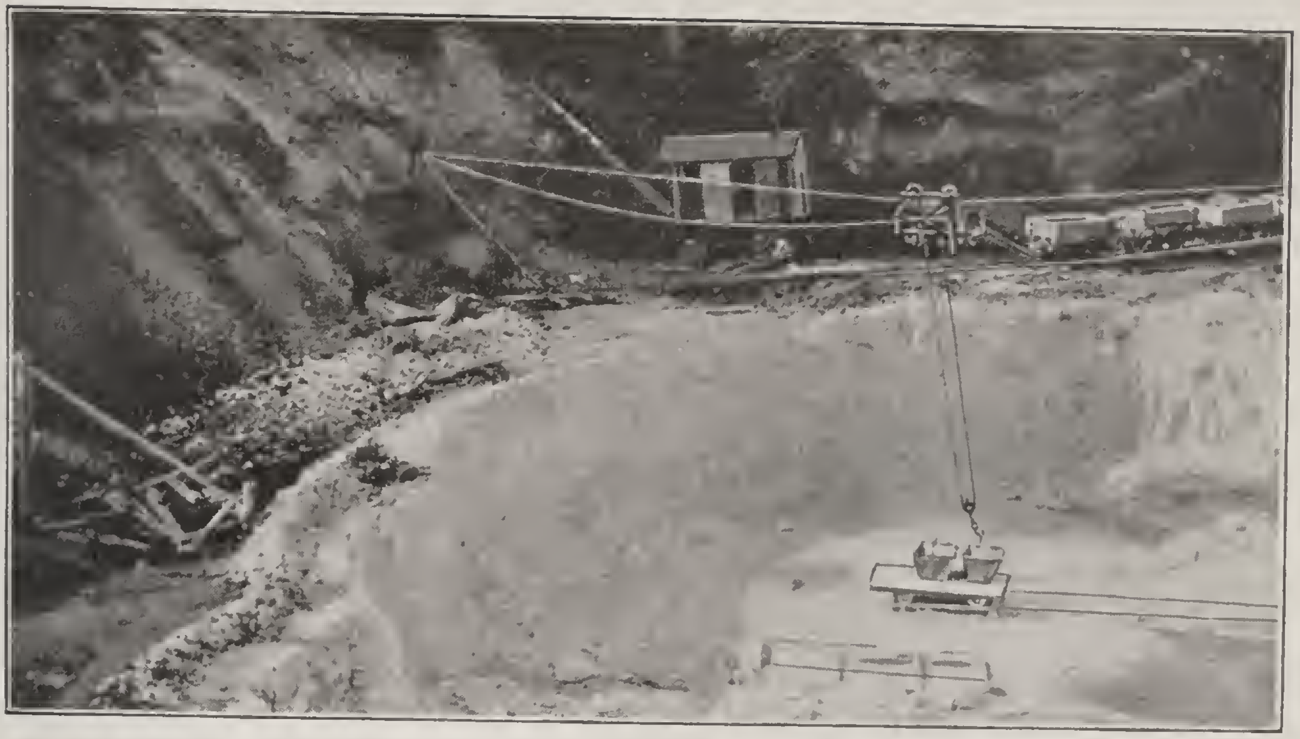
The John Sant Clay Co. mines and pulverizes its clay for use in the manufacture of white-ware pottery, wall tile, and other ceramic wares. The clay from this mine is very hard, fine grained, and burns to a good color, but lacks uniformity in composition, which is partly due to impurities. If this clay were to be refined it would doubtless find a much wider application in the ceramic industries.

ATLANTA MINING & CLAY CO.

The Atlanta Mining & Clay Co. formerly operated a mine and washing plant in this district. The property has been idle for the past year or more.

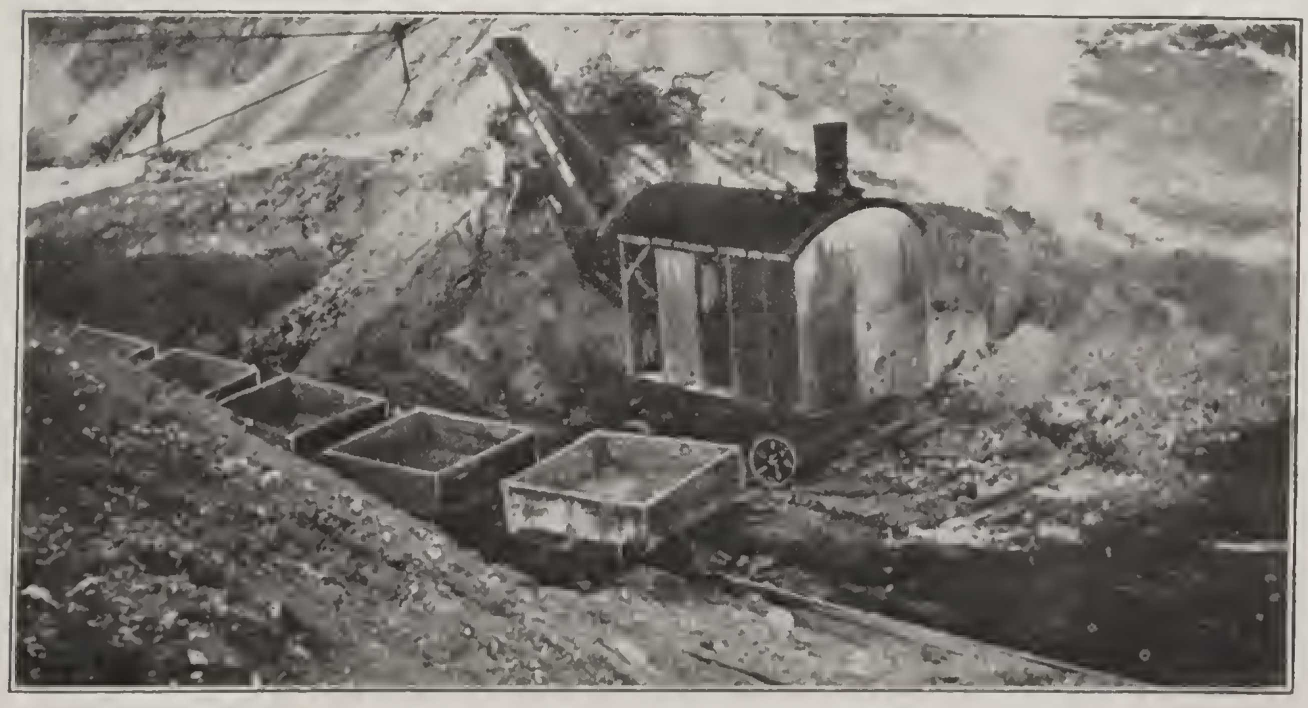
PROBABLE CAUSES OF VARIATION IN PHYSICAL PROPERTIES OF THE CLAYS.

The variation in physical properties of kaolin from different mines, as noted above, is evidently due to the variable conditions under which the deposits were laid down, the denser and harder clays having been formed under conditions of complete deflocculation and disintegration of clay grains, whereas the clays of lesser density and tensile strength have probably been deposited while the grains were in a more flocculated and coagulated condition. In other words, the extent of dispersion at the time of deposition has affected the physical properties of the clay. As a result the clay from one of the mines mentioned is hard and weak and must be finely ground before it will blunge satisfactorily with water; another mine produces soft mealy clay which is easily blunged with water without any preliminary treatment; and still another mine yields a soft, highly plastic, and strong clay which contains irregular layers or pockets of fine-grained flinty clay of low tensile strength, that is so hard that much of it must be thrown away, as it will not blunge in the washers. The harder clays contain less impurities than the softer ones, indicating that the ancient ocean or sea, which acted as a huge washing plant in refining and sorting these sediments, deflocculated some clays and caused the impurities to settle out more readily.



A. VIEW IN OPEN-CUT KAOLIN MINE IN THE DRY BRANCH DISTRICT, GA., SHOWING OVERHEAD TRAMWAY.

B. VIEW IN ANOTHER KAOLIN MINE, DRY BRANCH DISTRICT.



C. REMOVING OVERBURDEN WITH STEAM SHOVEL.

MINING METHODS AND TREATMENT.

KAOLIN MINING IN THE DRY BRANCH DISTRICT.

Open-cut mining is universally practiced at the kaolin mines in the Dry Branch district, owing to the fact that the overburden is so soft that any timbering would have to be water-tight in order to keep out sand and iron-stained silt in rainy weather.

REMOVAL OF OVERBURDEN.

Figure 1 shows graphically the strata in the mine of the Georgia Kaolin Co. and is typical of the strata generally found in mines of this district. However, the thickness of overburden will vary from about 10 feet along the ravines to probably 100 feet below the crest of the hill.

The removal of this overburden is an important part of the mining operations. To remove such large amounts of material economically and with dispatch and dispose of them has been one of the main problems. However, until lately not much attention has been given it.

At present (March, 1916) two of the companies mentioned are using steam shovels for removing the overburden, and it will probably be only a short time until the overburden is stripped back far enough (200 feet) from the clay face or breast to avoid contamination in case of caving during rainy weather. A steam shovel at work is shown in Plate I, *C*. Another company removes the overburden with pick and shovel, removing just enough to expose the face of the clay bank, with the result that sometimes after a rain the overburden caves, filling the pit and staining the clay. The drag scraper which was formerly used by two of the companies is shown in Plate II, *A*.

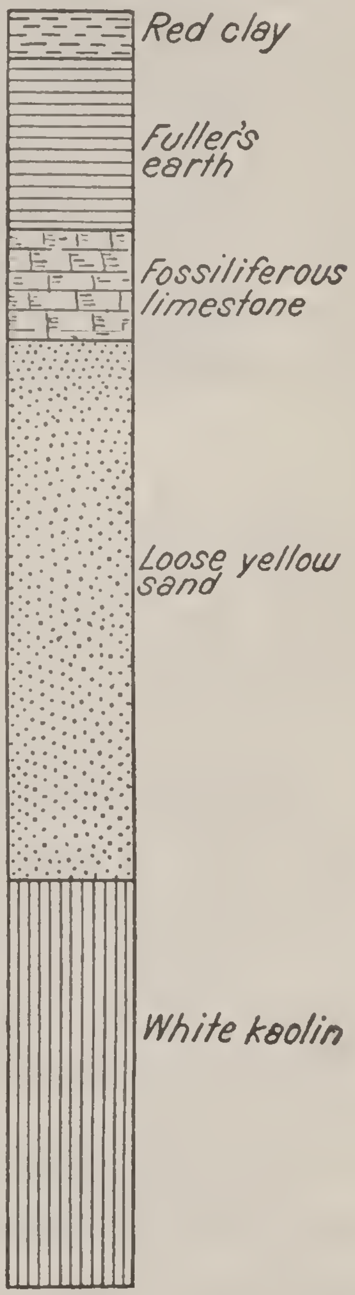


FIGURE 1.—Cross section of strata in a kaolin mine in the Dry Branch district.

MINING THE CLAY.

In the three mines mentioned the kaolin is mined with pick and shovel, the clay being mined economically and easily in this way, owing to the jointing. Veatch [a] in describing the mining of kaolin at the mine of the Georgia Kaolin Co., says:

> A structural feature which has an economic application is the jointing. The joints have no definite system or fixed direction, and may be oblique, vertical, or horizontal,

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[a] Veatch, Otto, Second report on the clay deposits of Georgia: Bull. 18, State Geol. Survey, 1909, p. 127.

and the bed may be minutely jointed or there may be only large cracks at wide intervals. The jointing has the appearance of being due to shrinkage of the clay mass, and not to orogenic movements.

By undermining the clay at the base of the face or breast and driving long wooden or steel stakes down into the clay near the edge of the breast, large chunks are split off, advantage being taken of the jointing of the clay. The clay split down from the bank is broken into lumps of a size that can be conveniently loaded. In the mines of the American Clay Co. and John Sant Clay Co., where the clay is not given a subsequent treatment other than grinding, these small lumps are hand sorted and the iron stains and sand pockets are cut out with hoe-shaped knives. The irregular jointing of kaolin in two different mines is shown in Plate II, *B* and *C*.

REMOVING CLAY FROM PIT.

Each company uses different methods in removing the mined clay from the pit, the method used depending upon the output of the mine, the topography of the surface, and the subsequent treatment of the clay.

JOHN SANT CLAY CO. MINE.

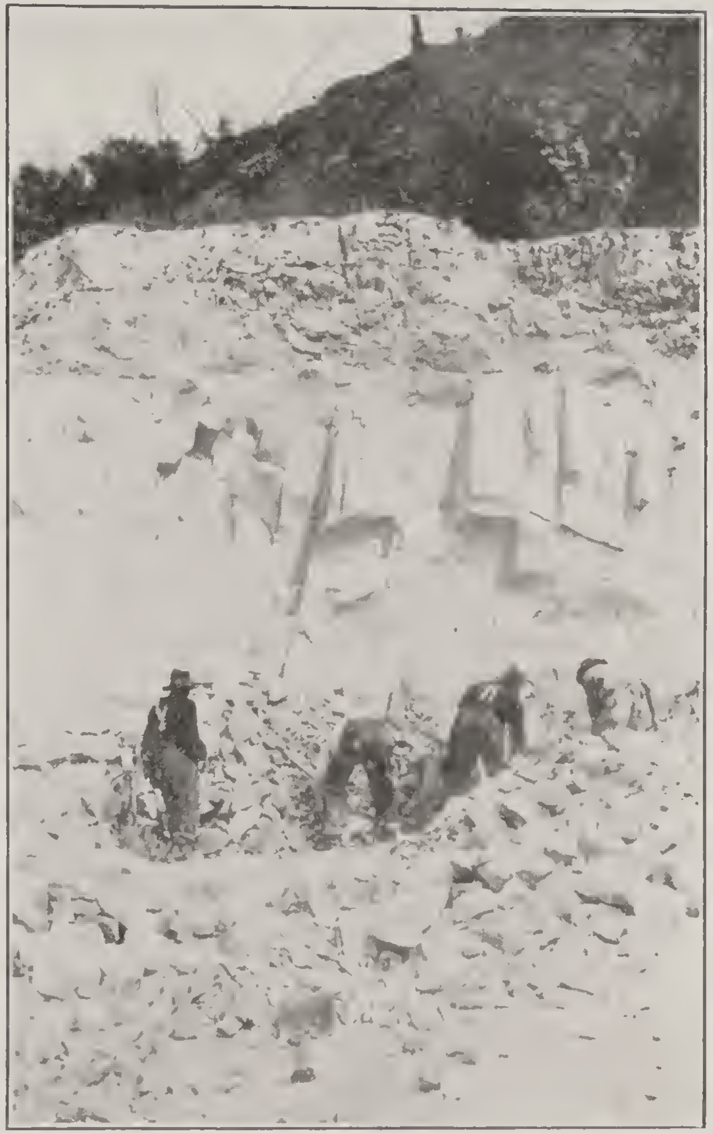
The mine of the John Sant Clay Co. is located about 1 mile from their drying sheds, which are situated along the Macon, Dublin & Savannah Railroad, and the output is small. The clay is shoveled into wagons and hauled to the open-air drying sheds where it is allowed to dry for several weeks. When dry the clay is ground in a set of corrugated rolls, screened, and sacked for shipment to the pottery trade.

The drying sheds (See Pl. III, *A*) consist of three to six tiers of superimposed racks about 15 feet wide, the clay being piled on the racks to a depth of about 3 feet. The bottom of the racks is formed of movable scantlings or pine railings. As the clay on the lowest tier becomes dry it is dropped on the floor, and the loads on each superimposed tier are dropped in succession. The dried kaolin dumped on the floor is broken with mauls or in roll crushers.

AMERICAN CLAY CO. MINE.

At the mine of the American Clay Co. the clay is elevated out of the pit with an overhead tramway which transports and dumps the clay into small loading bins at the top of the pit. The clay is then dumped into cars, which are drawn from the loading bins to the open-air drying sheds by a wire cable and winding drum. The clay is allowed to dry for several weeks, when it is shaken down to the bottom of the shed, hammered and cut up into small lumps about the size of a man's fist, loaded into cars, and shipped exclusively to the paper trade.

A. REMOVING OVERBURDEN WITH DRAG SCRAPER.



B. VIEW IN OPEN-CUT KAOLIN MINE, SHOWING IRREGULAR JOINTING OF CLAY.

C. ANOTHER VIEW SHOWING JOINTING OF CLAY.

GEORGIA KAOLIN CO. MINE.

In the mine of the Georgia Kaolin Co. the clay is merely broken in pieces small enough to be loaded by two men, as the subsequent treatment of the clay does not require breaking into small pieces at this stage. The clay is elevated from the pit by an overhead tramway or by cars which are pulled out of the pit by a steel rope and winding drum. Both the cars and buckets automatically dump their clay into a large storage shed, which holds enough clay to run the washing plant for a week.

KAOLIN REFINING IN THE DRY BRANCH DISTRICT.

Only one company, the Georgia Kaolin Co., in this district is actively engaged in the refining of kaolin. The method of washing clay at this plant is similar to the refining process used in refining the primary kaolin of the southern Appalachian district, which has been described in detail by Watts,[a] so only a mere outline of the washing process will be given here. It consists of seven different operations, which are as follows:

1. Thorough blunging of the clay with water, in rectangular blungers, which are about 10 feet long, 4 feet wide, and 4 feet deep.
2. Floating the slip through narrow troughs about 50 feet long with an average width of about 2 feet, the width of trough depending on the rate of flow required to settle out the impurities.
3. Screening the slip through a 110-mesh rotary screen.
4. Concentrating the slip in large concrete tanks, which are about 75 feet long, 25 feet wide, and 5 feet deep, where it is allowed to stand for about 48 hours, when the supernatant liquid is drawn off and the thickened clay slip run into a small retaining tank.
5. Filter pressing the thick clay slip, which is pumped from the small retaining tank into the filter press at a pressure of 100 pounds.
6. Drying the filter-press cakes, which contain about 25 per cent of water, in tunnel driers heated by exhaust steam, or in open-air drying sheds.
7. Crushing the thoroughly dried cakes into small pieces in corrugated rolls.

Open-air driers for drying the filter-press cakes are shown in Plate III, *B*. A view of the refining plant is shown in Plate III, *C*.

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[a] Watts A. S., Mining and treatment of feldspar and kaolin in the southern Appalachian region: Bull. 53, Bureau of Mines, 1913, p. 72.

IMPURITIES IN GEORGIA KAOLINS.

The following is a chemical analysis of a sample of kaolin taken from the pit of the Georgia Kaolin Co., as reported by S. W. McCallie:

Composition of sample of kaolin.[a]

	Per cent.
Moisture at 100° C	1.22
Loss on ignition	13.46
SiO_2	44.76
Al_2O_3	38.41
Fe_2O_3	.63
CaO	.20
MgO	.09
Na_2O	.09
K_2O	.35
TiO_2	1.37

From this analysis it is seen that the chief color-imparting oxides are Fe_2O_3 and TiO_2. The source of these oxides is best shown by the following results of a microscopic examination of a sample of clay from the mine of the Georgia Kaolin Co., by F. B. Laney, of the United States Geological Survey:

Associate minerals found by microscopic examination of kaolin.

Quartz.
*Partly decomposed feldspar, often stained with iron oxide.
*Wad—small pisolites of manganese oxide.
*Limonite.
Muscovite.
*Magnetite.
*Hematite.
*Ilmenite.
Zircon.
*Rutile.
Apatite.
*Tourmaline.
Corundum.
Monazite.
*Iron—minute particles broken off from the crushing machinery.

The minerals marked with an asterisk are the minerals that tend to give a cream color to a burned china body when Georgia kaolin is one of the constituents of the body.

Only a small part of these impurities is removed by the present methods of refining, owing to the fact that the most of the color-imparting minerals are present in a very fine state of subdivision, as the coarser impurities settled out when the clay was being deposited. In fact, most of the color-imparting particles will pass a 260-mesh sieve, and some are much finer. Besides being finely subdivided, these impurities are completely coated with clay, which prevents their being settled out, regardless of their higher specific gravity, without a large loss of clay. This clay coating is not removed to any great extent on prolonged blunging. Blunging for two hours will not disperse the clay particles. Grinding in a ball mill has a greater

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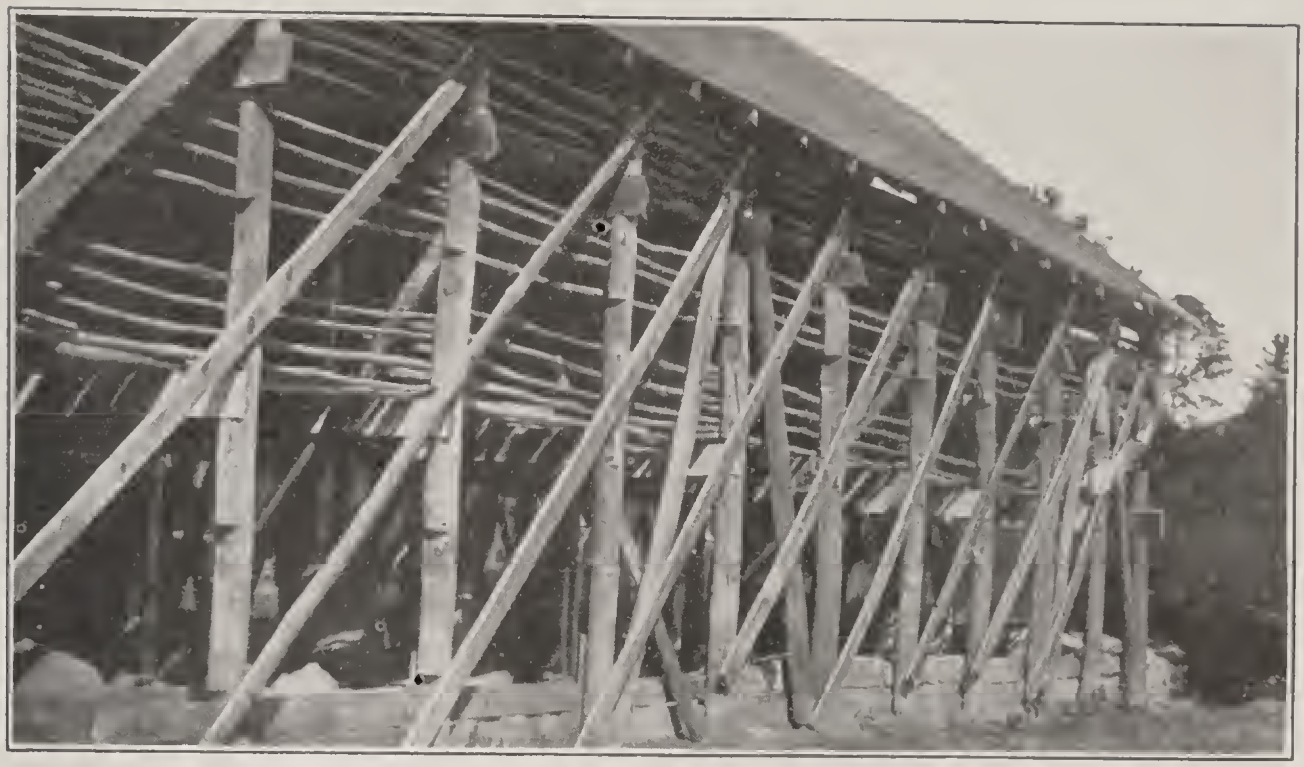
[a] Veatch, Otto, Second report on clay deposits of Georgia: Bull. 18, State Geol. Survey, 1909, p. 128.



A. OPEN-AIR DRYING SHEDS FOR CRUDE CLAY.

B. OPEN-AIR DRYING SHEDS FOR WASHED CLAY.

C. KAOLIN REFINING PLANT NEAR DRY BRANCH, GA.

tendency to produce complete disintegration, but such treatment is not economical to use in the washing of kaolins. Therefore, in order to obtain complete dispersion of the clay grains and the resulting separation of the coloring minerals from the clay it is necessary to consider the colloidal nature of clay, and apply chemical principles thereto.

EFFECTS OF ELECTROLYTES ON CLAY BODIES.

LITERATURE.

Probably one of the first persons who noticed the effect of alkalies on clay was Brongniart,[a] who in 1844 claimed that adding 3 per cent of potash improved the pottery bodies with which he worked.

Shortly after this, Hartmann [b] described the use of alkalies in casting.

Seger [c] also noticed some effects of alkalies in ceramic bodies.

Zebisch,[d] who carried on one of the first scientific investigations of the effect of bases on clays, noticed that basic substances have an angular sharp-cornered form while in a solution of water, and that acid substances have a round smooth form. He offers this as a possible explanation of the effect of bases on clays.

Goetz [e] in his German patent advises the use of soda with or without cinnabar in the casting of porcelain and stoneware.

In 1895 Hagedorn[f] claimed that a mixture of clay, chalk, quartz, and ground glass made a body requiring a smaller amount of water than certain other ceramic mixtures. This is probably due to the fact that alkalies dissolved from the ground glass and caused deflocculation of the clay.

Kosmann [g] claims that a certain amount of silicic acid is dissolved from the clay substance by soda, and that the silicic acid thus dissolved causes the clay slip to become more liquid.

New developments in colloidal chemistry about this time helped Lottermoser[h] to throw light on some of the present-day views of the effect of alkalies on colloidal gels.

Weber [i] claims in his patent that by using a plastic bond clay in a body together with soda, ammonia, potash, sodium silicate, molasses, and soap it is possible to cast with nonplastic ingredients a body as thick as 10 mm. in diameter. He states that such a slip will cast as well as the finer grained porcelain and stoneware slips.

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[a] Brongniart, A., Traité des arts céramiques ou des poteries, 1844, pp. 468–469.

[b] Hartmann, C., Die Tonwarenfabrikation (1850): Sprechsaal, Jahrg. 27, 1894, p. 1028.

[c] Seger, Hermann, Japanisches Porzellan und dessen Dekoration: Tonindustrie Ztg., Jahrg. 15, 1891, p. 813; Seger's Gesammelte Schriften, 1897, p. 573.

[d] Zebisch, A., Sprechsaal, Jahrg. 27, 1894, p. 105; Jahrg. 28, 1895, p. 303.

[e] Goetz, Karl, German patent 76247, 80b, Oct. 23, 1891; Ueber Sodaschlicher, Sprechsaal, Jahrg. 27, 1894, pp. 637, 685, 738.

[f] Hagedorn, Robert, and Hagedorn, Hermann, German patent 90697, Oct. 23, 1895.

[g] Kosmann, Dr., Ueber die Verflüssigung von Thonbrei durch Soda: Tonindustrie Ztg., Jahrg. 19, 1895, p. 382.

[h] Lottermoser, Alfred, Ueber anorganische Colloïde: Sammlung Chem. und chem.-techn. Vorträge, Bd. 6, 1901, pp. 161–240.

[i] Weber, E., German patent 158496, 80b, Aug. 14, 1902.

L. J. Briggs [a] and others, of the U. S. Bureau of Soils, used ammonia to dissolve the colloids in soils. It was found that ammonia was most effective in deflocculating soils containing a low percentage of soluble salts. For instance, little effect was noticed on soils containing more than 1 per cent of soluble salts. The investigators mentioned also noticed that when the soil contains a considerable amount of lime or magnesium carbonate, flocculation occurs, usually becoming more pronounced on the addition of ammonia.

Förster [b] notes that the viscosity of clays is decreased by small additions of alkalies. He claims that the negatively charged OH ion repels similarly charged clay particles. He also notes that the OH ion is absorbed by the clay. Spangenberg [c] claims that the OH ion is absorbed by the clay particles and the so-charged clay particles repel each other, causing a liquefaction of the clay slip.

Förster [d] claims that the Ca ion dissolved by clay slips from the plaster molds prevents to a certain extent the deflocculating action of the OH ion. In order to prevent this he advises the use of Na_2CO_3 instead of NaOH, thereby changing the Ca ion to harmless $CaCO_3$.

The German patent 201404 [e] issued in 1906, states that the plasticity of kaolin, clay, and ceramic mixtures can be increased by the addition of tannic acid. It also states that tannic acid with an excess of alkali when added to clay causes a certain fluidity in the slip, which on the addition of more tannic acid, again thickens. Certain other organic acids may be used in the place of tannic acid.

The introduction of alum [f] to accelerate sedimentation was another important step in the clay-washing industry. Hirsch,[g] Simonis,[h] Mellor,[i] Rieke,[j] and others studied the effect of soluble sulphates on clays. Most important is probably Rieke's work, in which he showed that $MgSO_4$, $CaSO_4$, $FeSO_4$, $ZnSO_4$, $CuSO_4$, $CoSO_4$, $Al_2(SO_4)_3$, and MgO increased the viscosity of a Zettlitzer kaolin slip.

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[a] Briggs, L. J., Martin, F. O., and Pearce, J. R., Centrifugal method of mechanical soil analysis: Bureau of Soils Bull. 24, 1904, p. 24.

[b] Förster, F., Ueber das Giessen des Tons: Chem. Ind., Jahrg. 28, 1905, pp. 733-740.

[c] Keppeler, Gustav, and Spangenberg, Albert, Verfahren Kaoline, Tone und aus hergestellte keramische Massen plastischer zu machen: Chem. Ind., Bd. 31, 1908, p. 592.

[d] Förster, F., Loc. cit.

[e] Keppeler, Gustav, and Spangenberg, Albert, German patent 201404, 80b, Aug. 29, 1906.

[f] Ries, Heinrich, Clays of the United States east of the Mississippi River: Prof. Paper 11, U. S. Geol. Survey, 1903, p. 35.

[g] Hirsch, H., Verhalten von Ton in Salzlosüngen: Tonindustry Ztg., 1904, p. 491.

[h] Simonis, M., Weitére Beiträge zum Verhalten von Tonen und Magérungsmitteln gegen Elektrolyte: Sprechsaal, Jahrg. 36, 1906, p. 1186.

[i] Mellor, J. W., Studies on clay slips: Trans. Eng. Cer. Soc., vol. 6, 1906-7, pp. 161-170.

[j] Rieke, Reinhold, Ueber die Wirkung löslicher Sulfate auf Kaoline und Tone: Sprechsaal, Jahrg. 43 1910, p. 709.

Alkalies were also used by Schwerin[a] in his electrical method of purification of clays. Before the clay slip was treated with electricity it was deflocculated with an electrolyte to permit the coarser impurities to settle. The finer impurities were then removed by electrolysis.

Ashley[b] found that the color of clays in which iron is one of the bases would be improved if the clay was blunged with hydrochloric acid, settled, and the overlying water drawn off.

Ashley[c] also reports Keppler's patent on improving clays by deflocculating with alkali, making an addition of colloid matter from humus, and reprecipitating all together by the addition of an acid.

Bleininger,[d] in summarizing his results on the electrical separation of clay, says the use of caustic soda and other electrolytes brings about a condition of minimum viscosity which greatly assists in the separation of quartz and feldspar from the clay.

Brown and Howat[e] found that by deflocculating kaolins with caustic soda or silicate of soda their color was improved and the drying shrinkage decreased, but the burning shrinkage increased.

Bleininger[f] states:

> The washing of kaolinitic clays may be decidedly improved as far as the quality of the product is concerned by adding to the water in the blunger a definite small amount of caustic soda or a mixture of caustic soda and sodium silicate. This treatment improves the color of most clays, enriches their content in clay substance, decreases the drying shrinkage, but tends to increase the burning shrinkage.

EXPERIMENTS WITH KAOLINS OF SOUTH CAROLINA AND GEORGIA.

PRELIMINARY INVESTIGATION.

The preliminary tests on the different secondary kaolins of South Carolina and Georgia were made by Robert Back in 1913–14 at the Washington laboratory of the bureau. Back investigated different methods of refining these secondary kaolins, and compared the color of the refined product when burned and glazed at cone 10 to that of English china clay burned under the same conditions. He found that by thoroughly deflocculating kaolin from the Dry Branch district with a small measured quantity of caustic soda the impurities could be settled out with greater ease. The product showed sufficient improvement in color when made up into hard, fire-glazed porcelain bodies to justify the washing of several tons of the Dry Branch (Ga.) kaolin at one of the washing plants operating in that district. The results of Back's work have been published by permission of the bureau in the Transactions of the American Ceramic Society.[a]

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[a] Stoermer, M., Ein neues elektrisches Tonreinigungsverfahren: Tonindustrie-Ztg., Jhrg. 36, 1912, pp. 1283–1284, Schwerin, —, Electrical process for the purification of clays; Trans. Eng. Cer. Soc., vol. 12, 1912–1913, pp. 36–65; Bleiniger, A. V., Use of sodium salts in the purification of clays and in the casting process, Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, pp. 14–19.

[b] Ashley, H. E., The technical control of the colloid matter in clays: Trans. Am. Cer. Soc., vol. 12, 1910, p. 802.

[c] Ashley, H. E., Op. cit., p. 803.

[d] Bleininger, A. V., Note on the electrical separation of clay: Trans. Am. Cer. Soc., vol. 15, 1913, p. 343.

[e] Brown, G. H., and Howat, W. L., The use of deflocculating agents in the washing of clays and the effect of the process upon the color: Trans. Am. Cer. Soc., vol. 17, 1915, p. 87.

[f] Bleininger, A. V., Use of sodium salts in purification of clays and in the casting process: Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, p. 38.

PURPOSE OF PRESENT TESTS.

In order to determine whether some practical method of refining the secondary kaolins of Georgia on a commercial scale could be devised, the Bureau of Mines arranged a cooperation agreement with the Georgia Kaolin Co., Dry Branch, Ga., and this work was begun by the writer in March, 1915.

EFFECT OF ALKALIES, ACIDS, AND SALTS ON GEORGIA KAOLIN SLIP.

Much work has been done in this country and abroad to determine the effect of electrolytes on clay in suspension in water, and in nearly every investigation carried on in this country Georgia kaolin was included among the clays tested. Therefore only a brief summary of the effect of alkalies, acids, and salts on Georgia kaolin suspensions, as determined in experiments by the writer, will be given here.

EFFECT ON VISCOSITY OF SLIP.

It is well known that alkalies in small amounts tend to increase the state of dispersion of clay-water systems, the result being a decrease in viscosity of the system. This breaking up of the larger clay particles into smaller ones is plainly shown by the following mechanical analyses of two samples of washed Georgia kaolin, one of which was washed in a solution containing 0.075 per cent NaOH. These analyses were made with a Schultze elutriation apparatus having an overflow of 80 c. c. per minute.[b]

TABLE 1.—*Results of mechanical analyses of washed Georgia kaolin.*

Size of clay particles.	Washed Georgia kaolin.	Georgia kaolin washed with 0.075 per cent NaOH.
Mm.	*Per cent.*	*Per cent.*
0.110	0.27	0.28
.0765	.42	.45
.0578	.33	.30
.0340	4.40	.92
.0187	2.36	2.30
.0100	15.70	9.04
	76.52	86.61

[a] Back, Robert, Effects of some electrolytes on clays: Trans. Am. Cer. Soc., vol. 76, 1914, pp. 515–546.

[b] For description of this apparatus see Watts, A. S., Mining and treatment of feldspar and kaolin in the southern Appalachian region: Bull. 53, Bureau of Mines, 1913, p. 46.

This table shows that the particles that were larger than 0.034 mm. were little affected by hydroxide; therefore they were chiefly impurities and not colloidal clay.

On the other hand, many acids and salts have the opposite effect to that of alkalies, causing a coagulation of the clay particles and a resulting increase in the viscosity of the system.

Therefore the effect of the addition of different amounts of electrolytes to a clay-and-water system can be satisfactorily studied by the determination of viscosity. This is best done by the use of a brass efflux tube or viscosimeter. The viscosimeter used by the writer was similar to the one described by Bleininger,[a] with the exception of the size of the efflux, which was $\frac{3}{32}$ instead of $\frac{5}{32}$ inch, as recommended by Bleininger. This reduction in size of efflux is necessary in determining the viscosity of slips containing a small percentage (less than 25 per cent) of solids. All viscosity determinations made by the writer are based on the volume of water that ran through the $\frac{3}{32}$-inch efflux tube in 117 seconds, which is to be understood when mention is made of viscosity determinations throughout this report. The viscosities of all clay suspensions are computed in terms of water by dividing their observed time of flow by 117.

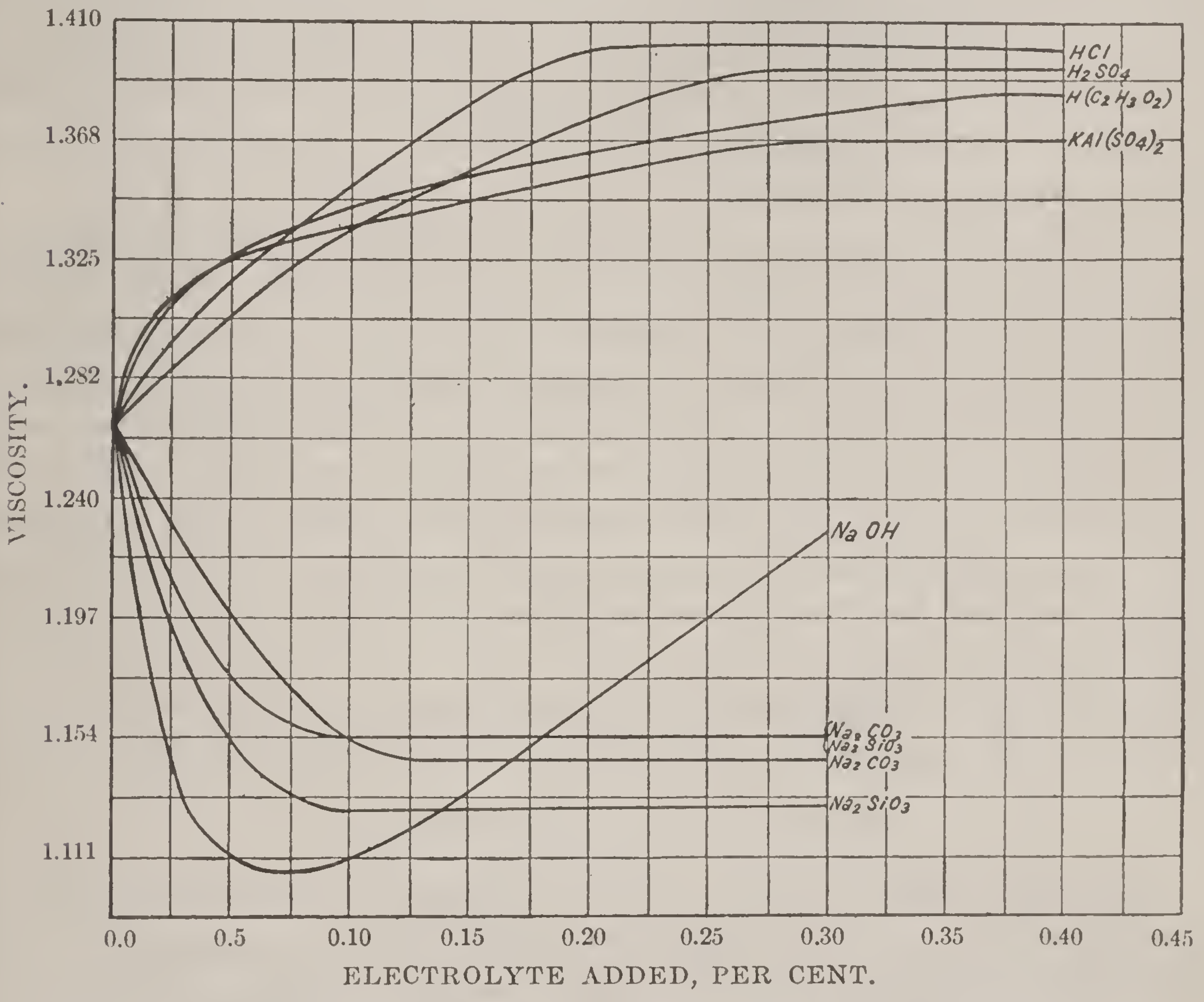

FIGURE 2.—Effect of electrolytes on viscosity of slip containing 20 per cent of Georgia kaolin.

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[a] Bleininger, A. V., Use of sodium salts in purification of clays and in the casting process: Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, p. 20.

Figure 2 shows the effect of alkalies, acids, and salts upon the viscosity of a Georgia kaolin slip containing 20 per cent clay. From these curves it is seen that caustic soda caused the greatest drop in viscosity of any of the reagents used, the viscosity reaching a well-defined minimum with an addition of 0.075 per cent of this reagent. On further additions of caustic soda the viscosity at once increased. In the case of sodium silicate and sodium carbonate the decrease in viscosity was not as great as that caused by additions of caustic soda. Also the minimum viscosity is not reached until 0.10 per cent of sodium silicate, or 0.125 per cent of sodium carbonate, is added to the clay-and-water system. A mixture consisting of 50 per cent Na_2CO_3 and 50 per cent Na_2SiO_3 caused a smaller drop in viscosity than either of these reagents alone, the minimum viscosity being reached with 0.10 per cent of the mixture. Neither sodium carbonate, sodium silicate, or a mixture of the two showed any tendency to increase the viscosity.

In the case of the coagulating agents hydrochloric acid showed the greatest coagulating tendency, the mixture reaching a well-defined maximum viscosity upon the addition of 0.2 per cent of the reagent. Acetic acid and alum did not increase the viscosity as much as hydrochloric acid, the maximum viscosity produced by these reagents being 1.384 and 1.376, respectively, as compared to 1.401 produced by hydrochloric acid; but they were much more active in coagulating the clay particles below 0.05 per cent additions. The addition of 0.4 per cent of acetic acid, or 0.3 per cent of alum, was necessary to reach the maximum viscosity.

In the case of sulphuric acid the maximum viscosity of 1.393 was reached with the addition of 0.275 per cent of the reagent.

EFFECT UPON SEDIMENTATION.

It is also a well-known fact that additions of small amounts of alkalies tend to hold the clay in suspension or in a state of deflocculation which corresponds to decreasing viscosity, while acids and salts have the opposite effect upon the slip, causing a more rapid settling of the clay particles. The slip is then said to be in a state of flocculation, which corresponds to an increase of viscosity.

CONTROL OF THE VISCOSITY OF CLAY SLIP.

The fact that alkalies when added to clay-and-water systems increase the dispersion of the system with resulting decrease in viscosity and increased suspension of the clay particles makes this principle directly applicable to the purification of Georgia kaolins, for the impurities in the Georgia kaolins are coated with clay, which fact makes it impossible to settle the impurities out without a great loss of clay, but when the clay-and-water system is dispersed they are then free to settle. Also the consequent increased suspension of the clay particles gives a longer time in which impurities may settle before the process becomes uneconomical owing to the settling of clay. The success of any kaolin-refining process making use of these facts depends upon the technical control of the viscosity of the clay-water system. Therefore a study of the factors affecting both the initial and the minimum viscosity was made under the conditions of a commercial washing plant.

FACTORS AFFECTING THE INITIAL VISCOSITY.

It is a well-known fact that present-day kaolin-refining methods are not only wasteful but very inefficient, which is shown clearly by viscosity tests performed by the writer at Dry Branch, Ga. These tests not only point out the cause of inefficiency, but show why better results have not been obtained.

VARYING CLAY CONTENT.

Without some mechanical means of controlling the amount of clay going into the washer, wide variations in the percentage of clay in the slip will occur. These variations are sometimes very great, as is shown in the table following, which shows the variation in clay-and-water content of the slip during a period of one month. From the data in this table it is seen that the percentage of clay varied from 0.6 to 12.7 per cent, with an average of 7.64 per cent for the month. These tests were made at all hours of the day and under varying conditions of weather, temperature, mining operations, and other changing factors, so that the average of 7.64 per cent of clay represents the average conditions found in the washing plant. It can be readily seen that this wide range in composition of slip will affect both the quantity and the quality of the product.

TABLE 2.—*Percentages of clay and water in slip during one month of operation.*

Date.	Hour.	Water.	Clay.	Date.	Hour.	Water.	Clay.
(1916)		*Per cent.*	*Per cent.*	(1916)		*Per cent.*	*Per cent.*
Mar. 20	9.00 a.m.	90.0	10.0	Apr. 7	8.15 a.m.	95.4	4.6
24	10.00 a.m.	91.6	8.4		9.45 a.m.	93.9	6.1
	11.00 a.m.	92.7	7.3		10.00 a.m.	96.0	4.0
25	9.00 a.m.	93.6	6.4		11.00 a.m.	94.4	5.6
	10.00 a.m.	90.2	9.8	8	6.45 a.m.	91.8	8.2
	1.00 p.m.	93.6	6.4		8.15 a.m.	88.9	11.1
26	8.00 a.m.	92.4	7.6		8.30 a.m.	94.2	5.8
	10.00 a.m.	93.6	6.4		8.45 a.m.	94.5	5.5
	12.00 a.m.	94.0	6.0		1.30 p.m.	91.5	8.5
27	8.00 a.m.	92.0	8.0	9	8.30 a.m.	88.9	11.1
	10.00 a.m.	91.8	8.2		10.00 a.m.	92.7	7.3
29	8.00 a.m.	94.8	5.2		2.00 p.m.	91.8	8.2
31	11.30 a.m.	95.6	4.4	10	8.00 a.m.	90.5	9.5
Apr. 1	8.00 a.m.	91.7	8.3		9.00 a.m.	87.3	12.7
2	9.30 a.m.	93.0	7.0		10.30 a.m.	93.9	6.1
3	2.45 p.m.	95.5	4.5		11.30 a.m.	89.4	10.6
5	7.30 a.m.	99.4	0.6	12	7.15 a.m.	86.5	13.5
	8.00 a.m.	92.2	7.8	14	6.30 a.m.	93.4	6.6
	9.00 a.m.	94.2	5.8		8.00 a.m.	91.4	8.6
	10.00 a.m.	88.7	11.3	15	5.00 p.m.	88.6	11.4
	1.30 p.m.	90.9	9.1	16	10.00 a.m.	89.6	10.4
	1.45 p.m.	94.7	5.3		10.30 a.m.	95.2	4.8
	5.00 p.m.	91.8	8.2		1.00 p.m.	91.8	8.2
7	7.15 a.m.	91.8	8.2		4.00 p.m.	91.2	8.8

Average, 7.64 per cent clay.

DEGREE OF BLUNGING.

The effect of different degrees of blunging on the initial viscosity is shown in curves *1*, *3*, and *4* of figure 3. Curves *1* and *3* were plotted from viscosity tests of slips taken direct from the washer or blunger of the plant at which the tests were made, the tests of curve *3* being made in dry weather while those of curve *1* were made in wet weather. From these two curves it is seen that there is a decided increase in viscosity of clay treated during wet weather. This increase is due to the fact that the clay is wet and sticky as it enters the washer, which in turn decreases the blunging action of the washer, causing a decrease in the dispersion of the clay particles with consequent increase in viscosity. It is a well-known fact at kaolin-refining plants that the clay will not settle as quickly in wet weather as in dry weather, and alum is used to precipitate the clay. This fact seems to be contradictory to the above statements, but the reason for the slow rate of settling is not due to increased dispersion but to the decreased percentage of clay in the slip.

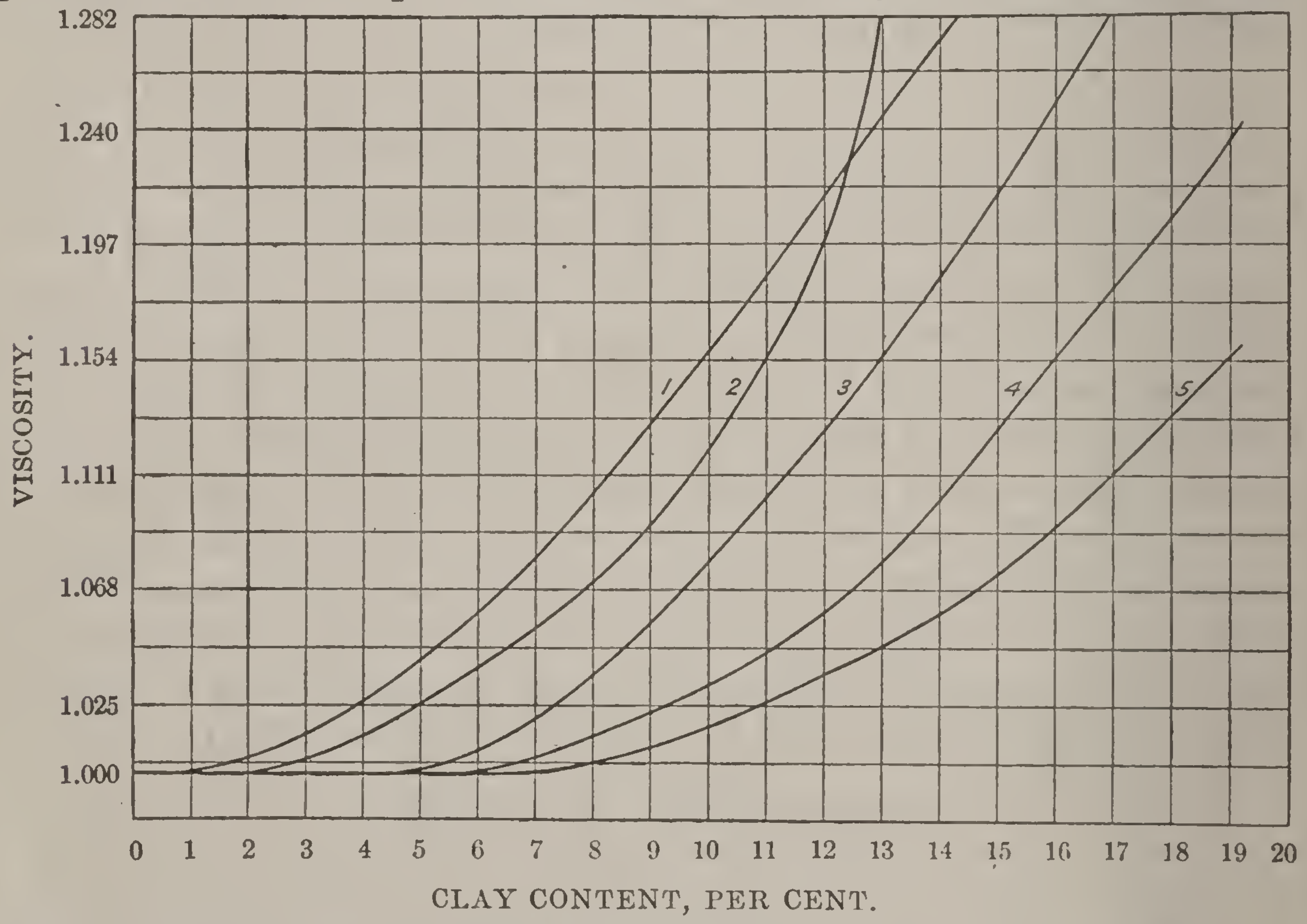


FIGURE 3.—Effects of moisture in clay used, blunging, and additions of alum, on Georgia kaolin slip. *1*, test made during wet weather; *2*, water contained 0.03 per cent alum; *3*, test made during dry weather; *4*, undried clay blunged 2 hours in ball mill; *5*, bone-dry clay blunged in small churn.

The effect of increased blunging action is shown in curve *4* of figure 3. The slips for this test were made from clay taken direct from the mine during dry weather, blunged in a ball mill for two hours, screened through a 120-mesh sieve, and then tested. From this curve it is seen that increased blunging causes an increased drop in the viscosity, owing to increased dispersion.

PREDRYING.

Curve *5* of figure 3 shows the effect of drying clay before blunging with water. These tests were made by blunging bone-dry clay with water in a small household churn. The curve shows that drying produces still greater dispersion with resulting low viscosity.

USE OF ALUM.

The tests represented by curve *2* of figure 3 were made in dry weather, but the feed water contained about 0.03 per cent of alum. From this curve it is evident that small percentages of alum will greatly increase the viscosity, the effect of the alum becoming more pronounced as the percentage of solids in the slip increases.

THE EFFECT OF VARYING VISCOSITY ON CAPACITY OF WASHING PLANT.

The capacity of a washing plant depends on the maximum flow of water through the washer and the percentage of clay held in suspension at the maximum viscosity at which precipitation of impurities is not retarded.

This maximum permissible viscosity will vary to some extent with clays from different deposits and can be determined only by close observation of a number of settling tests of slips of varying viscosities. In most cases a viscosity of about 1.094 will be the maximum. However, it is advisable to run at a viscosity a little lower than the maximum, a viscosity of 1.068 being safe practice.

The maximum amount of water that will flow through the ordinary rectangular washer, 10 feet long, 4 feet high, and 4 feet wide, without overflowing and produce maximum dispersion under given blunging conditions depends on the condition of the clay entering the washer, whether wet or dry, lumpy or disintegrated. Under the present-day conditions of kaolin refining, where the clay goes into the washer in large lumps, the maximum flow is about 1,200 cubic feet per hour.

It has been shown that a number of factors affect the percentage of clay contained in a slip of a given viscosity. Therefore these factors tend to increase or decrease the capacity of a washing plant, according to whether they decrease or increase the viscosity of the slip. For example, suppose the plant is operating under dry-weather conditions and maintaining a constant viscosity of 1.094; the slip will then contain 10.8 per cent of solids, but under wet-weather conditions the slip will contain only 7.4 per cent of clay, a decrease in capacity of 31 per cent, whereas if the clay has first been dried the slip will contain 16.2 per cent solids, an increase in capacity of 50 per cent over dry-weather conditions.

FACTORS AFFECTING MINIMUM VISCOSITY.

As the previous tests showed that caustic soda produces the greatest dispersion in Georgia kaolin slip, it was used in commercial tests of kaolin refining, and it was found that not all of the factors affecting the initial viscosity had an effect on the degree of dispersion or the minimum viscosity produced by small additions of alkalies.

DEGREE OF BLUNGING.

It is seen upon comparing the curves of figure 4, plotted from tests made in dry weather, with those of figure 5, plotted from tests made in wet weather, that the degree of dispersion or minimum viscosity of slips having the same clay content is the same, regardless of the wide difference in initial viscosity. For example: From the curve in figure 4, representing a slip which contained 9.6 per cent of clay, it is seen that the initial viscosity is 1.059 and the minimum 1.025, whereas the curve shown in figure 5, representing a slip containing 9.4 per cent solids, shows an initial viscosity of 1.145, but the same minimum of 1.025.

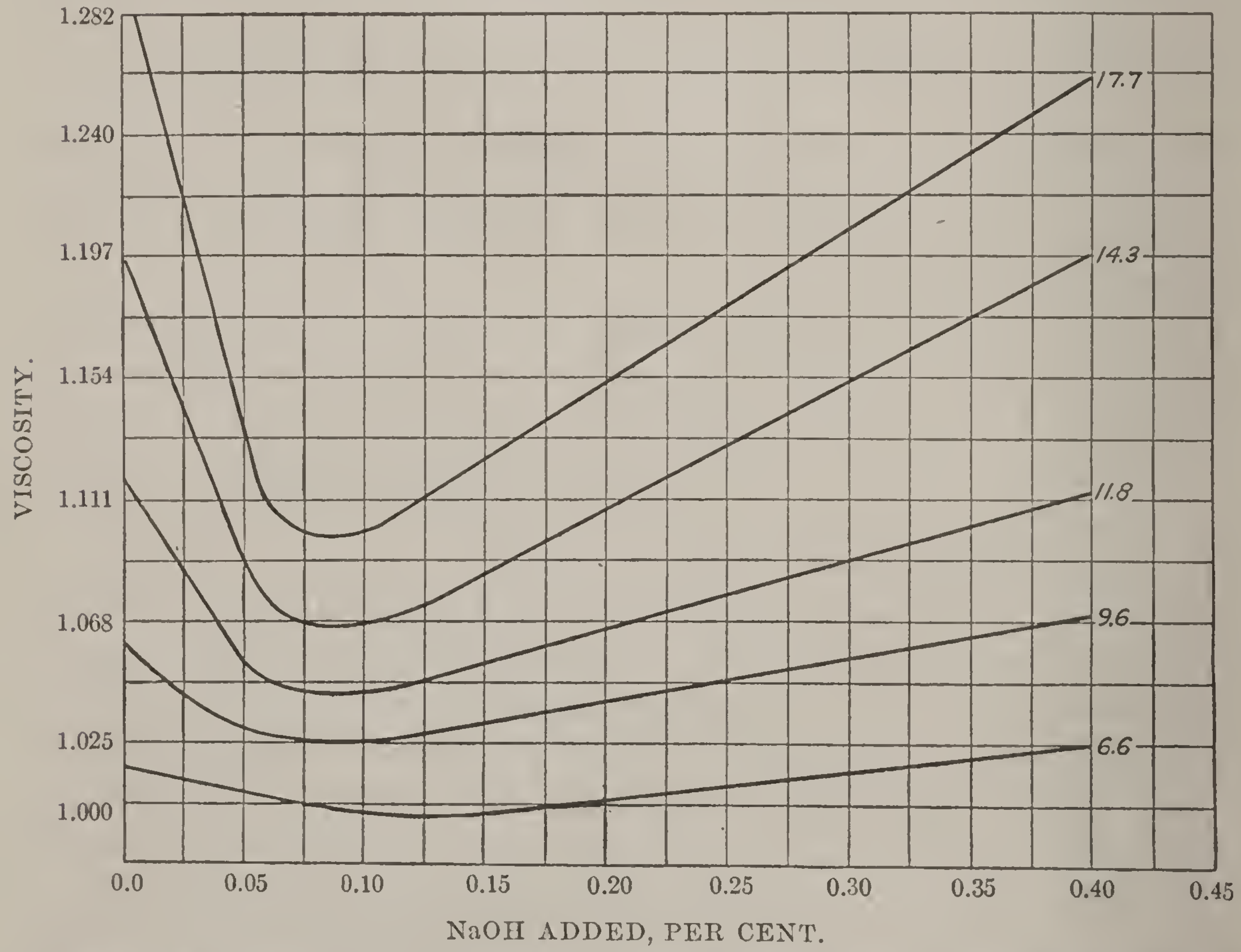


FIGURE 4.—Effect of caustic soda on viscosity of Georgia kaolin slips under dry-weather conditions. Figures on right show percentage of clay in slip.

Comparison of results shown by the curves of figure 6, which were plotted from tests of slips blunged in a ball mill, with those of the curves shown in figure 4, indicates that similar conditions prevailed. Although there was a large drop in initial viscosity, the minimum viscosity is the same for slips containing the same percentage of clay. Therefore the amount of blunging has no effect on the degree of dispersion or minimum viscosity of a slip of given clay content.

PREDRYING.

The curves shown in figure 7 were plotted from tests with slips of Georgia kaolin which had been previously dried to "bone dryness" on the top of steam pipes. By comparing these curves with those of figure 4, it is seen that predrying the clay decreases the initial viscosity—in slips containing less than 15 per cent clay—to that of the minimum viscosity. In other words, when the slip contains 15 per cent or less of clay, predrying is just as effective in increasing dispersion as additions of caustic soda, but predrying is far more expensive than using the electrolyte. However, on adding caustic soda to such slips the viscosity is decreased still further, but the drop is very small.

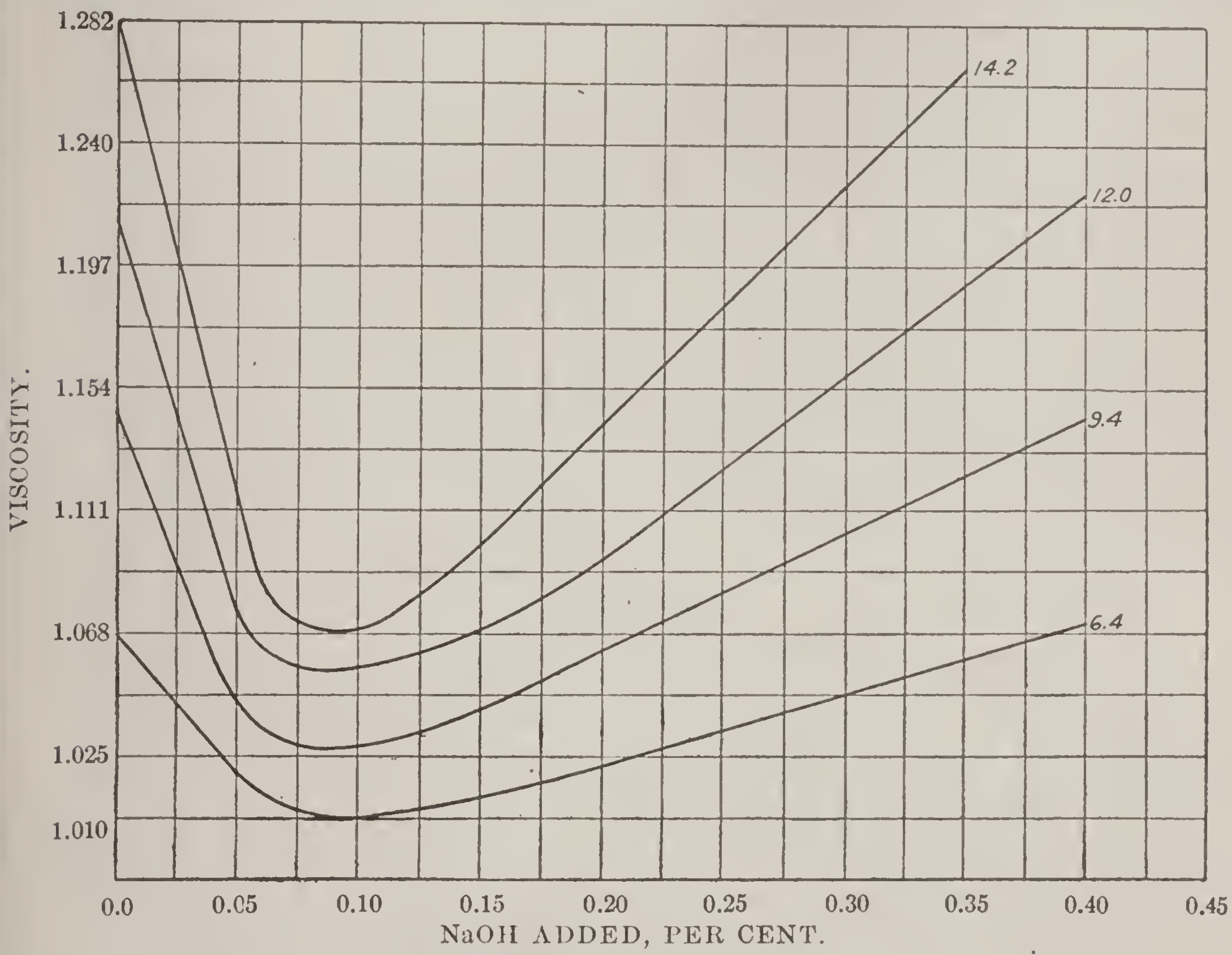


FIGURE 5.—Effect of caustic soda on viscosity of Georgia kaolin slips under wet-weather conditions. Figures on right show percentage of clay in slip.

As the minimum viscosity is not changed by the amount of blunging, and only slightly by severe predrying conditions, the capacity of a plant using a dispersing agent will be constant under such changing operating conditions.

USE OF ALUM.

The results of tests made with slips containing a small percentage (0.03 per cent) of alum are shown by the curves in figure 8. By comparing these curves with those of figure 4 it is seen that alum raises the initial viscosity, increases the amount of caustic soda necessary to produce minimum viscosity, and decreases the degree of dispersion. Therefore the presence of a flocculating agent such as alum in the feed water is very objectionable to a successful application of the technical control of disperse systems in kaolin refining.

REFINING KAOLIN BY TECHNICAL CONTROL OF DISPERSE SYSTEM.

SEDIMENTATION PROCESS.

From laboratory experience and from suggestions made by Ashley, Bleininger, and others, it appears that the best method of utilizing caustic soda as a deflocculating agent in kaolin refining is as follows:

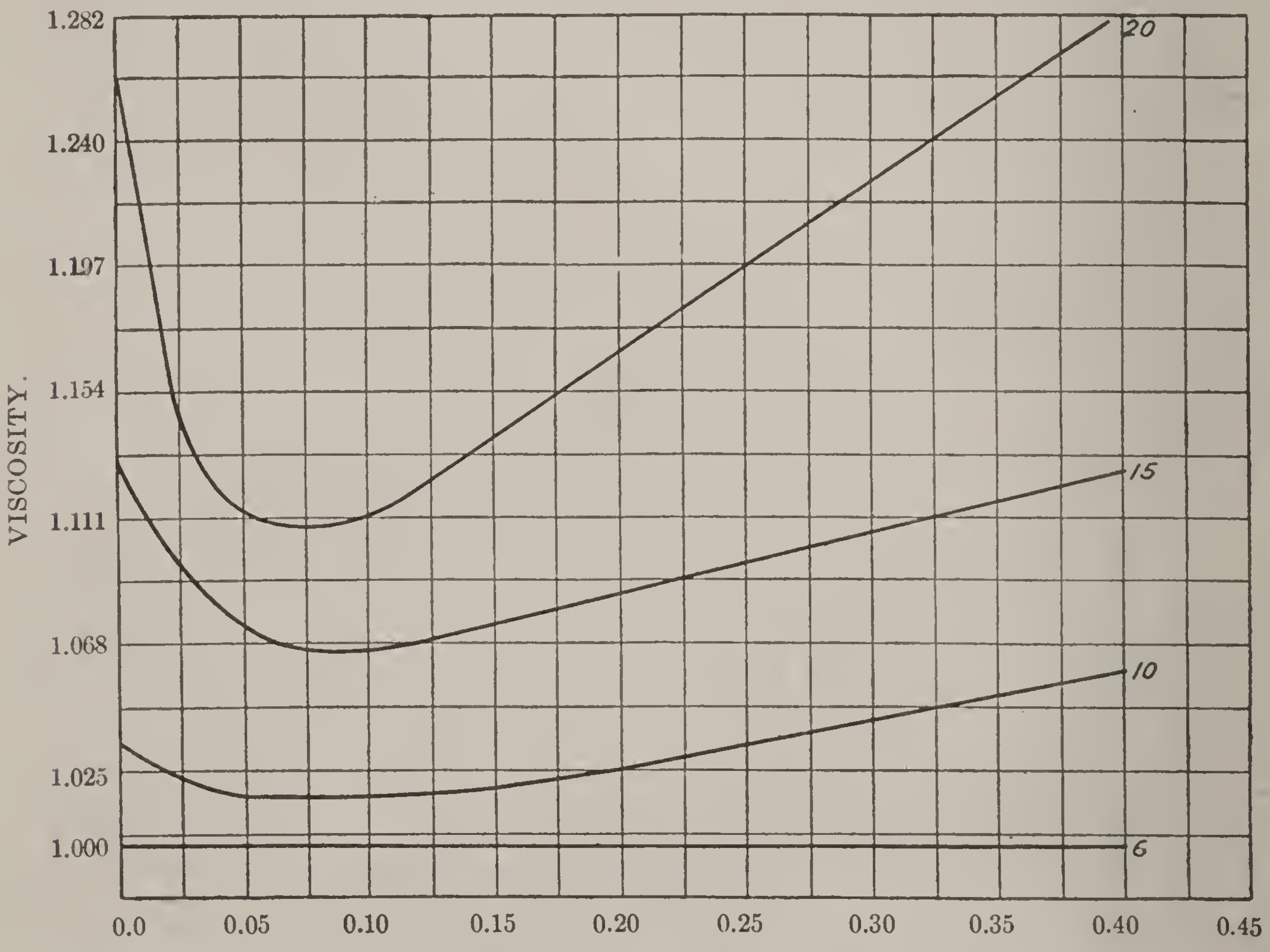


FIGURE 6.—Effect of caustic soda on viscosity of Georgia kaolin slips blunged in ball mill.

Sufficient caustic soda to cause maximum deflocculation or minimum viscosity is added to the clay and water in the washer. The deflocculated slip is run through the settling troughs, screened, and collected in the concentration tank, where it is allowed to stand long enough for the impurities to settle out. The slip is then siphoned into another tank and a flocculating agent added in order to coagulate the clay particles and concentrate the slip for filter pressing.

However, the results of a number of tests conducted in a small concrete tank 6 feet square and 2 feet deep showed that settling out the impurities in large tanks was not an economical process. The time required to settle out the impurities through a large quantity of slip was comparatively long, being about two hours, and in the meantime much clay had also settled, which had to be thrown away with the impurities. Also several inches of slip had to be left in the bottom of the settling tank when the slip was drawn off to prevent some of the impure material being sucked up. Aside from the large loss of clay caused by use of such a method, the continuity of the refining process is stopped, which adds to the operating costs.

THE ELUTRIATION OR SLIMING PROCESS.

Experiments on settling out the impurities in troughs were next carried on. A new set of troughs was built for this purpose and after a number of changes in width and length of the last trough, good results were obtained. The elutriation process consists of 11 different operations which are as follows: (1) Storage and disintegration, (2) feeding, (3) blunging, (4) dispersion, (5) sliming, (6) coagulation, (7) screening, (8) concentration, (9) filter pressing, (10) drying, and (11) crushing.

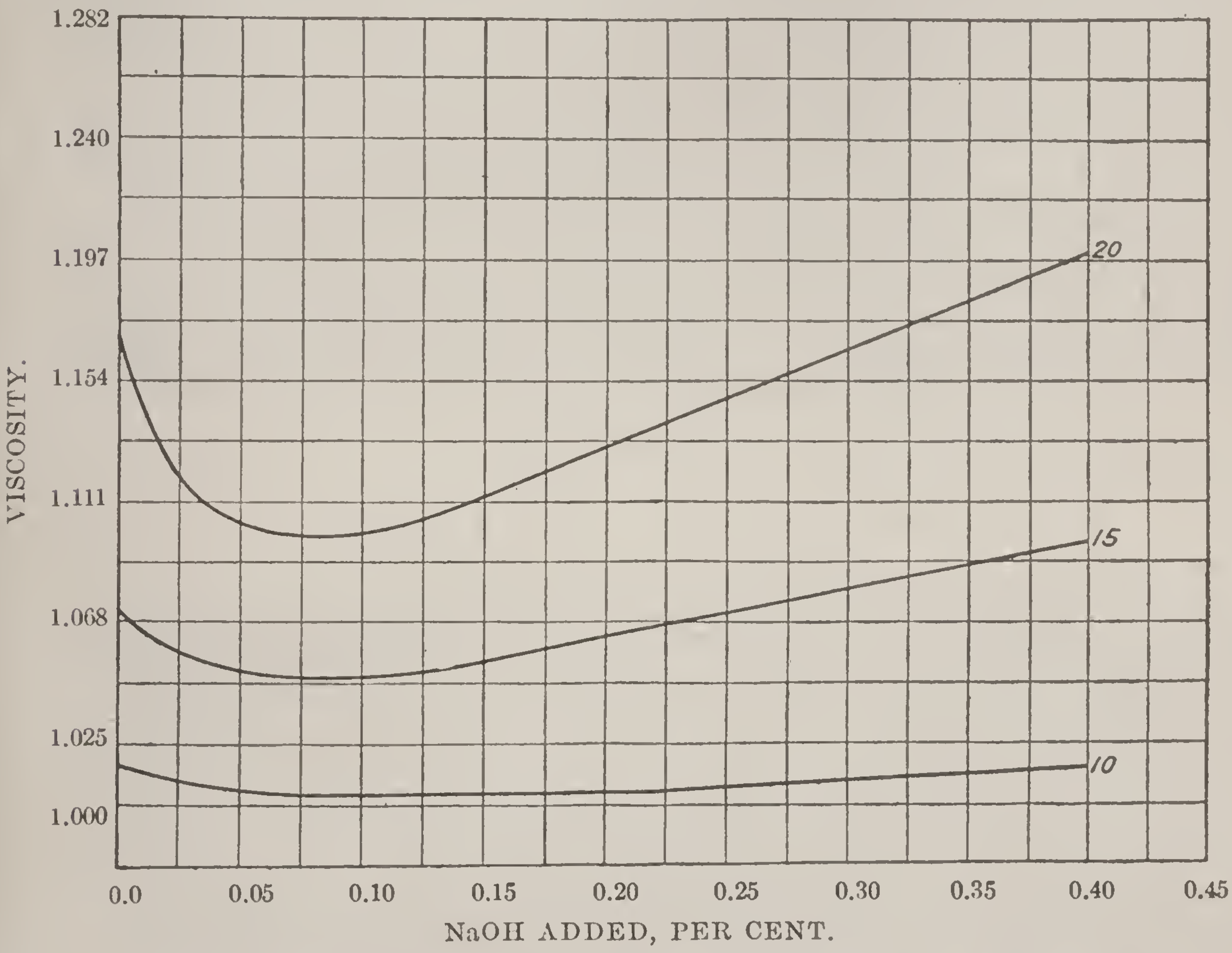


FIGURE 7.—Effect of caustic soda on Georgia kaolin slips made from predried clay.

STORAGE AND DISINTEGRATION.

The clay should be broken into lumps no larger than a man's fist and most of the clay should be much finer. In order to disintegrate the clay in this manner it is necessary to have a storage shed of at least 500-ton capacity, because during rainy weather the clay as mined from the pit is too sticky to be broken up.

All washing plants should have a storage shed for the crude clay regardless of the method of washing employed, for it has been shown that the capacity of the plant will vary greatly with the condition of the clay entering the washer unless it is dispersed with an alkali. However, few kaolin refining plants have a storage shed; therefore the writer wishes to emphasize the fact that the first thing to be considered in constructing a washing plant, if constant conditions are to be maintained with resulting uniform product, is a well-constructed storage shed of sufficient capacity. The disintegrator should be of the centrifugal roll type, as this type works best with highly plastic clays.

FEEDING.

It has been shown that the quantity of caustic soda required to produce maximum dispersion depends upon the percentage of clay in the slip; therefore it is expedient to have a constant flow of clay into the washer. The best way to obtain a constant flow is by the use of a clay feeder of the disk type. No feeder will deliver clay to a washer with any uniformity unless the clay has been previously crushed into small lumps; hence the necessity of using a disintegrator. One type of feeder is shown in Plate IV, *A*.

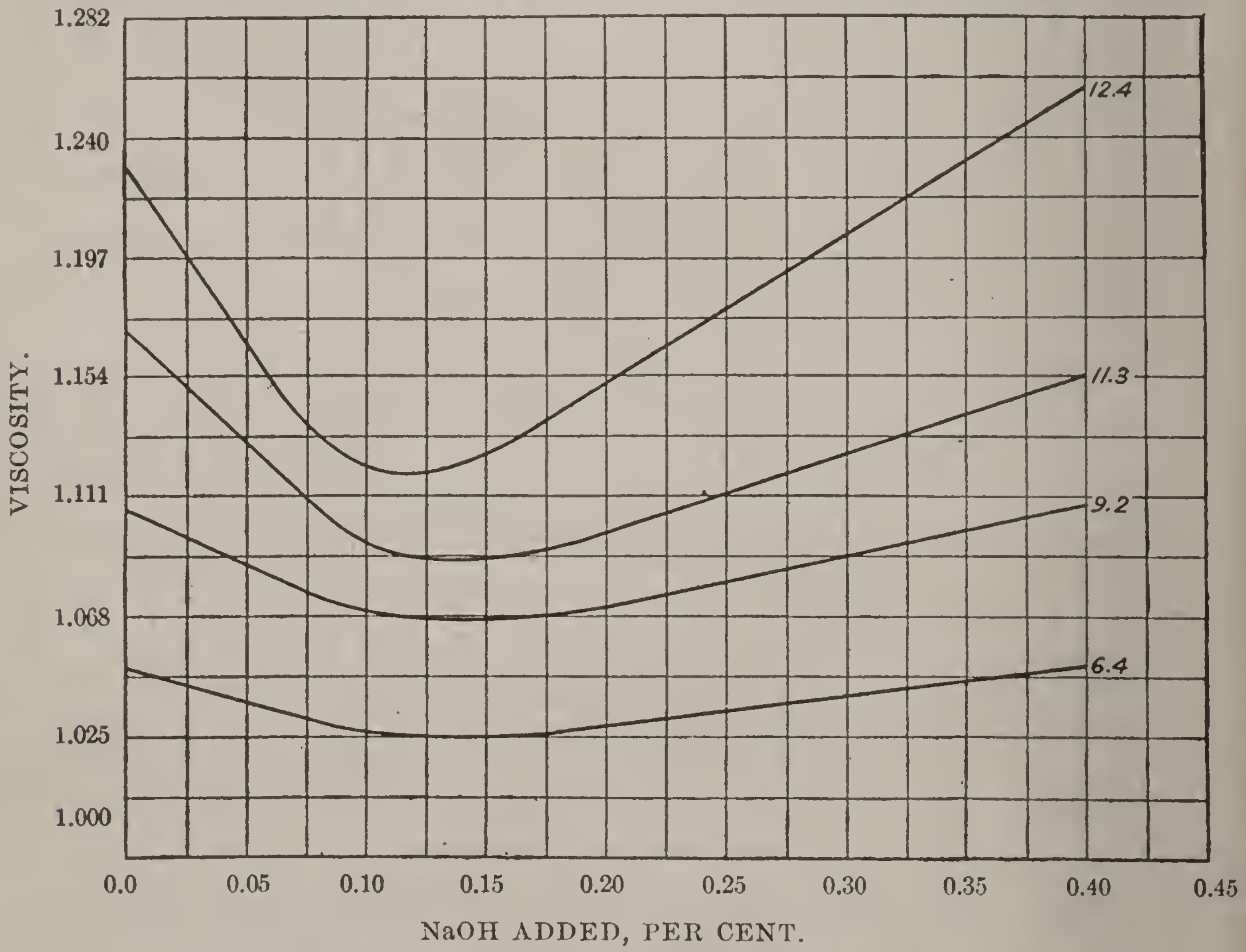
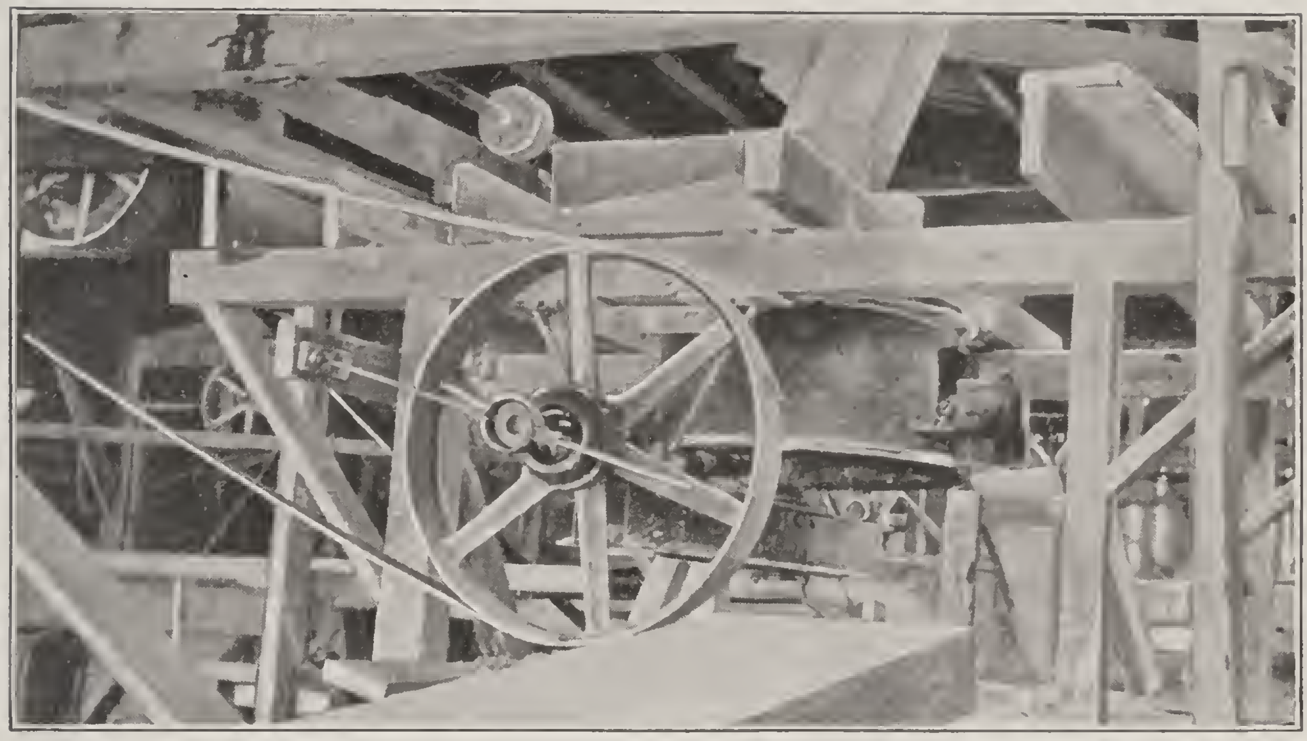


FIGURE 8.—Effect of caustic soda on Georgia kaolin slips containing 0.03 per cent of alum.

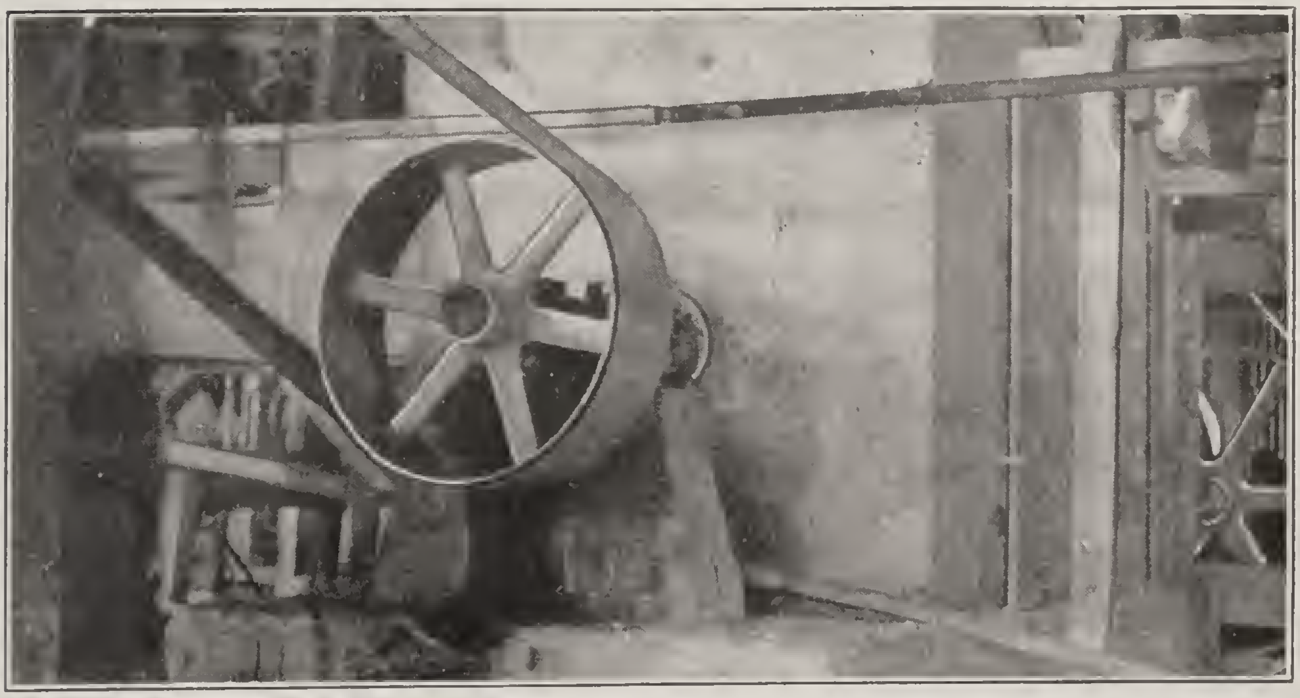
BLUNGING AND DISPERSION.

Blunging and dispersion will be discussed together as they are very closely related.

Blunging the clay, water, and alkali is one of the most important steps in the washing process, and too much emphasis can not be



A. DISK CLAY FEEDER.



B. CONCRETE WASHER OR BLUNGER.

laid upon this operation. Double blunging or washing is necessary to obtain complete dispersion and in the case of some very tough and tenacious clays that resist blunging it will be necessary to use three washers in order to get complete separation of the clay particles and impurities.

It is advisable to add the caustic soda to the slip in the second blunger, because if the flow of clay or water should be increased or decreased the clay content of the slip can be calculated from viscosity or specific-gravity tests of the slip leaving the first washer, and the flow of standard caustic-soda solution can be so regulated as to produce maximum dispersion under the changed flow of clay or water. In case the maximum permissible viscosity is exceeded the process should be stopped. In order that there will be no serious stoppage of the flow of clay or water into the washer there should be a storage bin of about 8-ton capacity, placed above the clay feeder, and the water fed into the washer by a No. 5 centrifugal pump. This pump should be set below the level of the bottom of a large concrete water-supply tank, about 30 by 15 by 8 feet, so that the water will run directly into the pump and prevent it from sucking air, which necessitates priming the pump.

TYPES OF WASHERS.

The rectangular washer used in nearly all washing plants has proved very successful in blunging the clay, but the power consumption has been very high, owing to faulty construction. The concrete washer illustrated in figure 9 and shown in operation in Plate IV, *B* will give efficient results, and the cost of repairs on this type of washer will be much lower than on the rectangular wooden box ordinarily used, as it can be easily relined without the necessity of rebuilding the whole washer.

ADDITION OF CAUSTIC SODA.

A standard solution of caustic soda is made up, and this solution is run into the second washer at a constant rate of about 10 cubic centimeters per second, which is obtained by allowing the solution to run through an efflux tube one-eighth of an inch in diameter. This efflux tube is placed about 2 inches from the bottom of the caustic soda supply tank so that it will not be easily clogged by any material that has settled to the bottom of the tank. The supply tank should also be equipped with two auxiliary efflux tubes of smaller diameter for regulating the quantity of caustic soda if the flow of clay is changed. The tank, which should have a capacity of 150 gallons, should be made from alkali-resisting material, such as enameled steel, and should contain an agitator. The head or top of the tank should be made air-tight in order that the effect of the varying head due to the lowering of the level of the solution can be overcome by the application of Mariotte's principle of the flow of liquid from an immersed tube, a constant flow of standard solution into the washer being thus obtained.

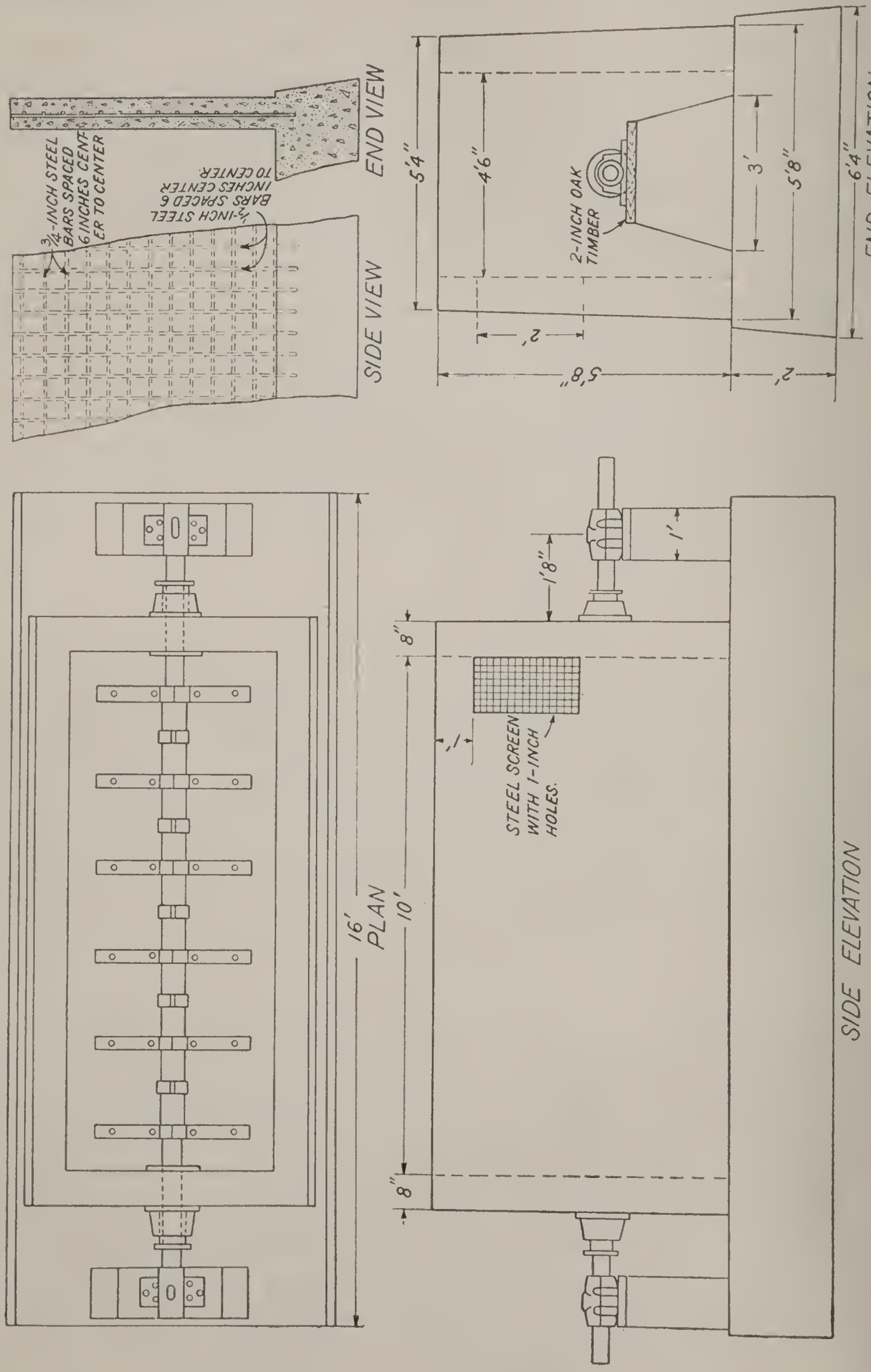

END VIEW
SIDE VIEW
3/4-INCH STEEL BARS SPACED 6 INCHES CENTER TO CENTER
1/2-INCH STEEL BARS SPACED 6 INCHES CENTER TO CENTER
5'4"
4'6"
2-INCH OAK TIMBER
3'
5'8"
6'4"
END ELEVATION
2'
5'8"
2'
1'8"
1'
8"
STEEL SCREEN WITH 1-INCH HOLES.
1'
16'
PLAN
10'
8"
SIDE ELEVATION

FIGURE 9.—Design of concrete washer.

The figures given above for the rate of flow of standard caustic soda solution and for the size of the main efflux tube are somewhat arbitrary and can be changed to suit different conditions.

STRENGTH OF CAUSTIC-SODA SOLUTION.

The proper strength of caustic soda solution to use will depend on the rate at which the solution is allowed to flow into the washer and the rate at which the clay is fed to the washer. For example, if 5 tons of clay is being fed to the washer every hour, then $5 \times 2000 \times 0.00075$, or 7.5, pounds, of caustic soda will have to be added every hour to produce complete dispersion. Now, if the rate of flow of the solution is 10 cubic centimeters per second, every cubic centimeter of standard solution will have to contain $\frac{7.5 \times 454.5}{3600 \times 10} = 0.094$ gram of caustic soda.

Therefore the normal solution is $\frac{0.094 \times 1000}{40} \mathrm{N} = 2.35\mathrm{N}$.

SLIMING.

After thorough blunging and complete dispersion the slip is run through the elutriation troughs. The troughs used by the writer with good success do not differ from the ordinary mica troughs used at present in most washing plants, except that a long fan-shaped trough is added at the end of the series of troughs.

This trough should be at least 125 feet long, 2 feet wide at the inlet, and the width at the discharge end should be such as to produce a depth of slip of not over 1 inch. The trough should be constructed with great care, for the bottom must be perfectly level both lengthwise and crosswise and the sides very smooth, as too great a frictional resistance to the flow causes coagulation of the slip along the sides and results in faster flow through the center of the trough. The bottom of the trough should be rough and uniform in texture, because on a smooth bottom the slip tends to roll the impurities along, and on an uneven bottom the slip tends to thicken around the spots of most resistance, causing it to flow too fast in other parts of the trough. The best material of which to construct these troughs is concrete, and by proper sizing of the aggregate and proper troweling the required surfaces can be obtained.

There will be less tendency for the clay particles to coagulate around spots of greater resistance if the slip is agitated before it enters the last trough. This can be easily done by having the fan-shaped trough lower than the others and then running the slip over riffles as it passes from the narrow troughs into the fan-shaped one. (See fig. 10.)

COAGULATION.

As the slip comes from the troughs it is still in a state of maximum deflocculation and therefore can not be concentrated as the exceedingly fine particles would not settle even after weeks of standing.

Furthermore, any attempts at filter-pressing the slip as it comes from the sliming troughs would be futile for, aside from the fact that the quantity of water to be removed would be so great that the process would not be practicable, the alkaline condition of the slip with the resulting dispersion of the clay particles would make filter-pressing almost impossible, as the exceedingly fine particles would tend to clog the filter cloth and what water did run through would be very turbid. Therefore it is necessary to add some flocculating or coagulating agent. Alum is usually used for this purpose, but in case the washing plant is using the water from the concentration tanks and filter presses over and over again alum can not be used, as a very small proportion (0.02 per cent or less) in the feed water decreases the effect of caustic soda and consequently increases the amount of soda

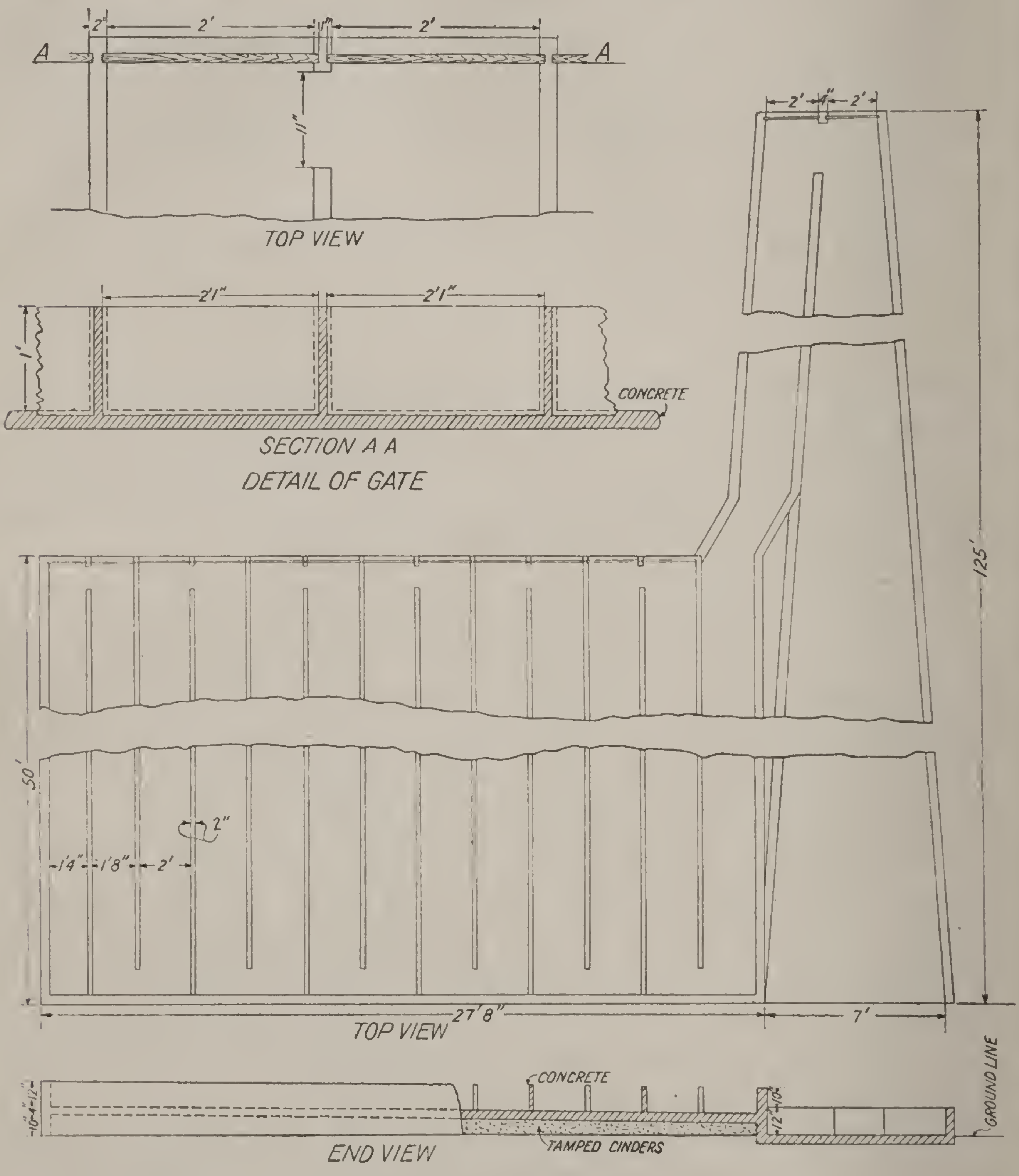


FIGURE 10.—Design of floating troughs.

necessary to cause maximum dispersion. If just enough sulphuric acid is added to neutralize the caustic soda, the slip is brought back to its normal state.

The sulphuric acid solution should be the same strength as the caustic soda solution, and it should be added to the slip in the same manner. Under these conditions perfect neutralization will result and there will not be an excess of caustic soda left to cause trouble in settling and filter-pressing or an excess of sulphuric acid left to cause trouble in blunging and dispersion. The sulphuric acid solution is added to the slip at the point where it leaves the sliming troughs. After the addition of the acid the slip is run through a long narrow trough, say 50 feet by 1 foot by 2 feet, containing a number of riffles for mixing the acid solution with the alkaline slip. The bottom of this trough should have a drop of at least 1 foot in 50 in order to cause a rapid flow.

SCREENING.

After the slip is neutralized by adding sulphuric acid it is advisable to run it through a rotary screen of about 120 mesh. This screen will remove any chips of wood or other foreign materials that float on the top of the slip; it also acts as an agitator in case some of the alkaline slip has not been completely neutralized. Care should be taken that practically all of the acid has combined with the caustic soda before reaching the screen, as any excess of acid will act upon the screen cloth and shorten its life.

CONCENTRATION.

The concentration of the slip is best accomplished by running the neutral slip into large concrete tanks (see fig. 11) 60 feet long, 30 feet wide, and 6 feet deep, where it is allowed to stand long enough for the clay to precipitate; then the supernatant liquid is drawn off, the time of settling governing the thickness of the slip. However, after about 96 hours the amount of precipitation is exceedingly small and in general 72 hours is sufficient for the concentration of the slip. This concentrated slip is then run into a small retaining tank, or, better still, is pumped into the filter presses direct from the concentration tanks.

FILTER-PRESSING.

Under present conditions of kaolin refining the filter-pressing of the thickened clay slip has given more trouble than all the rest of the washing operations. The troubles have been numerous and varied, and have been partly due to the type of filter press used, poor or clogged filter cloth, too thin a slip, condition of clay particles, whether dispersed or coagulated, and other factors. In view of these facts

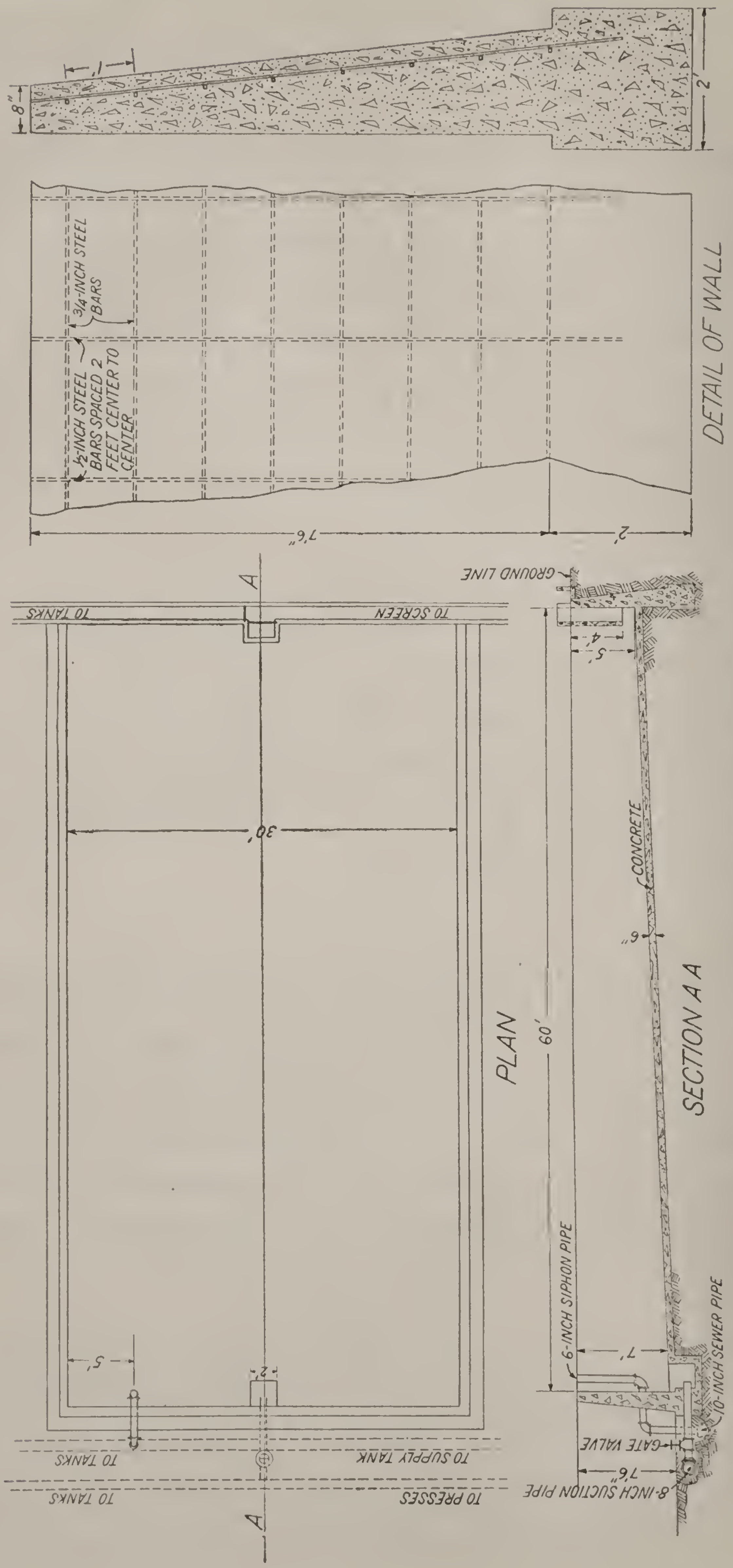


FIGURE 11.—Design of concentration tank.

numerous attempts have been made to devise other methods of drying the clay, but to the writer's knowledge none has proved more efficient than the filter press. The osmose purification process used in England [a] eliminates the filter press, but it is doubtful whether the apparatus would meet conditions in this country.

Bleininger [b] says:

> If the electrical continuous separation can replace the filter press economically its use would be justified. To a large extent it would be a matter of the comparative economy of the electrical endosmose apparatus and the filter press. If the latter should be improved, as has been done in the so-called clamshell press, it is quite possible that the economic advantage might remain with the filter press.

One of the chief sources of trouble in filter-pressing highly plastic clay slips is a slip that has not been sufficiently concentrated. In other words, the clay has not been allowed time to settle, owing, perhaps, to lack of tank capacity. The slip for filter-pressing should contain at least 30 per cent of clay. It must be remembered that in order to force this thick clay slip into the presses the pumps must be below the bottom of the tank from which they are drawing the slip; as the ordinary filter-press pump will not successfully lift a thick clay slip.

DRYING.

The drying of the filter-press cakes is best accomplished in a tunnel drier heated by exhaust or live steam. During the summer months open-air driers can be used to good advantage.

CRUSHING.

The filter-press cakes after being thoroughly dried are crushed into small pieces in a set of rolls.

LAYOUT OF PLANT.

The plan of a washing plant equipped for using the chemical process is shown in Plate V.

COST OF THE CAUSTIC SODA AND SULPHURIC ACID TREATMENT.

The extra cost over present-day methods of the caustic soda and sulphuric acid refining of kaolins may be divided into three factors: (1) First cost, maintenance, and operation of extra mechanical equipment required; (2) cost of chemicals; (3) chemist's salary. These costs, on the basis of 50 tons of clay treated per day, follow.

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[a] Ormandy, W. R., British clays under the osmose purification process: Trans. Eng. Cer. Soc., vol. 13, 1913-1914, pp. 35-48.

[b] Bleininger, A. V., Use of sodium salts in purification of clays and in the casting process: Tech. Paper 51, Bureau of Standards, Sept. 25, 1915, p. 19.

MECHANICAL EQUIPMENT.

The extra equipment necessary for the successful operation of the process is: One disk clay feeder, one washer, extra settling trough, and two 200-gallon tanks. The cost of operating and maintaining this extra equipment will be about $0.12 per ton of clay treated.

CHEMICALS.

With caustic soda at $0.03 per pound and sulphuric acid $0.025 per pound, the cost of chemicals will be $0.0825 per ton of clay.

CHEMIST.

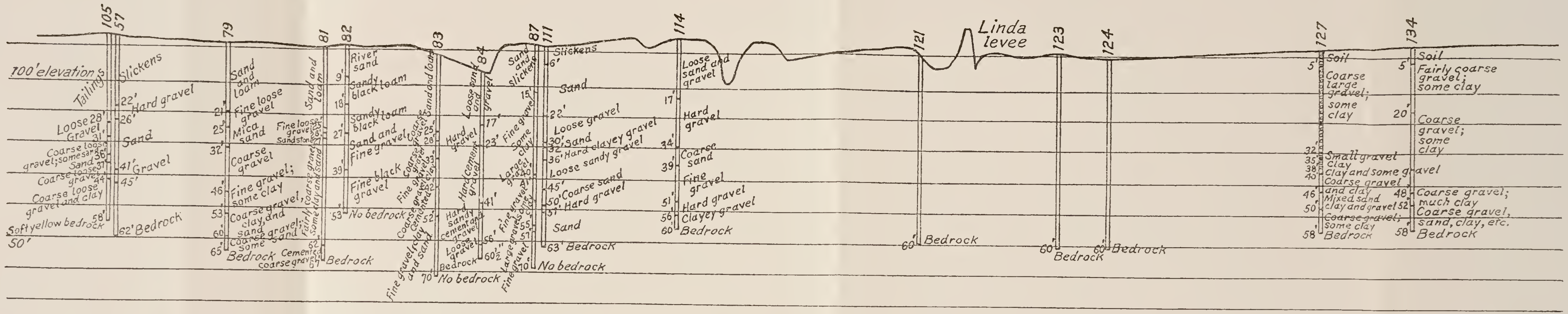
Successful operation will depend upon careful technical control of every step of the process. It will therefore be necessary to have a trained chemist available at all times. The services of such a chemist will approximate $0.15 a ton.

The extra total cost will therefore be about $0.35 a ton. However, it must be remembered in comparing the cost of the new with the old method that the caustic soda and sulphuric acid process increases the capacity of the plant. For example, slip with a maximum permissible viscosity of 1.094 contains after complete dispersion 17 per cent of clay, whereas the slip that is not dispersed contains 10.8 per cent during dry weather and 7.6 per cent during wet weather. As the average percentage of clay content, under the present-day methods, is 7.64 per cent, the increase in capacity will be 123 per cent, which will more than offset the increased cost of operation.

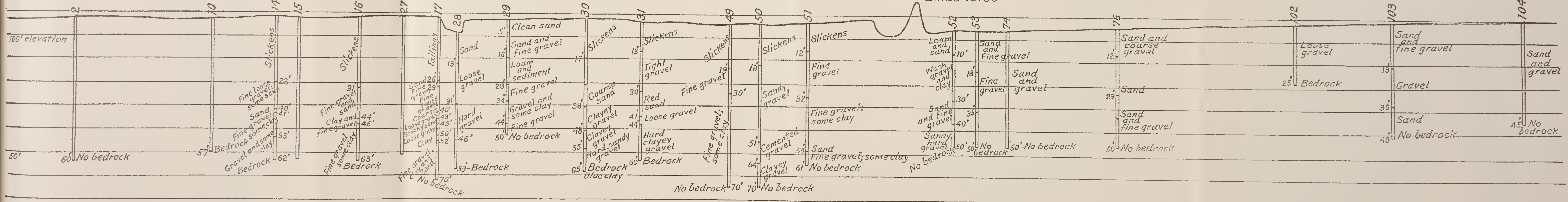
CONCLUSIONS DRAWN FROM RESULTS OF TESTS.

The application of the new process of treatment to kaolin refining on a commercial scale is thoroughly practical and the product has a purity impossible to attain under present methods of refining. In order to utilize the caustic soda sulphuric acid process with clay such as that experimented with it must be remembered that:

1. The clay and water must be fed into the washer at a uniform rate.
2. Double blunging or washing is necessary. Without thorough blunging the process is doomed to failure.
3. The caustic soda must be added in definite quantities, the amount depending upon the character of the clay and the quantity being fed to the washer.
4. A constant viscosity of about 1.068 must be maintained at the head of the sliming troughs in order to obtain a uniform product. The viscosity should never exceed 1.094.
5. It is advisable to agitate the slip before it is allowed to run through the fan-shaped trough.



Cross Section 5



Cross Section 6

SECTION SHOWING LOGS OF DRILL HOLES IN DREDGING GROUND.

6. The last sliming or settling trough should be fan-shaped, at least 125 feet long, 2 feet wide at the inlet, 7 feet wide at the discharge end, and perfectly level both lengthwise and crosswise. The sides of this trough should be very smooth; the bottom should be comparatively rough but uniform in texture.

7. Sufficient sulphuric acid to neutralize the caustic soda should be added, and no more.

8. It is absolutely necessary to have a chemist, one familiar with clay disperse systems, present at all times, for it is impossible to make the process fool-proof.

UTILIZATION OF REFINED GEORGIA KAOLIN.

After an improved process of kaolin refining has been worked out, the question, "Will the treated clay be sufficiently improved so that it can be utilized in the manufacture of the finer grades of ceramic wares by substituting it for the English china clay?" at once arises. In order to answer this question several tons of clay was refined by the new process at the plant of the Georgia Kaolin Co., to be used for test purposes. Owing to the fact that during the refining of this clay the flow of clay into the washer was not uniform because a clay feeder had not been installed and the amount of clay had to be approximately determined by viscosity tests taken every five minutes, it was not as pure as would be obtainable under standard conditions. However, on making a number of settling tests it was thought that sufficient improvement had been made to warrant some preliminary tests of the burned color of the clay.

PRELIMINARY TESTS TO DETERMINE PHYSICAL PROPERTIES AND COLOR.

SHRINKAGE.

The average linear shrinkage of the washed Georgia, treated Georgia, and English china clays are given in the table following. The shrinkage was determined by measuring the dry length and burned length of a 5-centimeter mark placed upon 10-centimeter disks in the green state. These disks were made by pressing the plastic clay into plaster molds. The disks were dried at 110° C. and the dry lengths measured, then fired in saggers at cone 10 in the kilns of the Mayer China Co., and the burning shrinkage determined.

Drying shrinkage and burning shrinkage of kaolin disks.

Kaolins tested.	Drying shrinkage, per cent.	Burning shrinkage, per cent.
English china clay	2.32	4.48
Washed Georgia kaolin	4.38	8.12
Treated Georgia kaolin	3.24	8.16

The table shows that the new process decreases the drying shrinkage about 25 per cent, but the burning shrinkage is about the same.

COLOR IN WHITE-WARE BODIES.

Some preliminary tests on the use of the treated Georgia kaolin in white-ware bodies were made in the ceramic laboratories of the Ohio State University, Columbus, Ohio. The purpose of these tests was to determine how much improvement in burning color the treated clay would show over that of the washed Georgia kaolin when made into glazed white-ware bodies, and also the approximate percentage of treated Georgia clay that could be substituted for the English china clay in a white-ware body without impairing the color of the glazed body.

MATERIALS USED IN MAKING BODIES.

The commercial materials used in this investigation were as follows:

Feldspar.—Canadian feldspar, Pennsylvania Feldspar Co.
Flint.—Pulverized quartz sand, Ohio Silica Co.
Florida kaolin.—Edgar Plastic Kaolin Co.
English china clay.—Moore & Munger's M. W. M. No. 1.
Ball clay.—Tennessee No. 9, Johnson & Porter Co.
Washed Georgia kaolin.—Georgia Kaolin Co.
Treated Georgia kaolin.—Georgia Kaolin Co.
Whiting.—Gilders.

The analyses of these materials are given in Table 3 following:

TABLE 3.—*Results of analyses of materials used in tests.*

Constituent.	Feldspar.	Flint.	Florida kaolin.	English china clay.	Ball Clay.	Washed Georgia kaolin.[a]	Whiting.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Silica (SiO_2)	65.15	99.18	45.50	47.37	51.20	45.53	1.30
Alumina (Al_2O_3)	18.65	.20	39.30	37.85	32.10	37.97	2.14
Iron oxide (Fe_2O_3)	.10	.06	.05	.31	.73	.73	.10
Titanium oxide (TiO_2)	.00	.01	.00	.04	1.40	1.55	.00
Lime (CaO)	.18	.01	.00	.00	.13	.05	54.40
Magnesia (MgO)	.00	.01	Trace.	Trace.	.00	.13	.00
Potash (K_2O)	10.36	....	.42	1.40	.69	....	.00
Soda (Na_2O)	4.31	....	.44	.14	.10	....	.00
Ignition loss	.32	.11	14.20	12.60	13.75	13.69	.00
Carbon dioxide	.00	.00	.00	.00	.00	....	42.68
Total	99.87	99.58	99.91	99.71	100.10	99.65	100.62

[a] The treated Georgia kaolin was not analyzed.

COMPOSITION OF THE BODIES USED.

The composition of the seven bodies that were compounded from the above white-burning materials is given in Table 4.

TABLE 4.—*Composition of bodies used.*

Material.	Proportion used in body No.—						
	1	2	3	4	5	6	7
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Feldspar	15	15	15	15	15	15	15
Flint	38	38	38	38	38	38	38
Florida kaolin	7	7	7	7	7	7	7
Tennessee ball clay	3	3	3	3	3	3	3
Whiting	2	2	2	2	2	2	2
English china clay	35	23	12	0	23	12	0
Washed Georgia kaolin	0	12	23	35	0	0	0
Treated Georgia kaolin	0	0	0	0	12	23	35
	100	100	100	100	100	100	100

PREPARATION OF THE BODIES.

The materials for each body, consisting of 8,000 grams, were weighed out and the total weight checked by weighing the batch. Each body was ground wet for five hours in a porcelain-lined ball mill, sieved through a 120-mesh screen, and divided into two parts, one part being used for casting small cups, and the other part being dried to a plastic state in plaster molds lined with cheesecloth, and after the clay had been wedged small pots like flower pots were made from it on a jigger.

PREPARATION OF TEST PIECES.

The bodies were made up into 48 test pieces each—24 by casting and 24 by molding on a jigger. These test pieces were first air-dried and then the drying was completed in a gas-fired drier.

BISCUIT BURN.

In order that the range of vitrification of the test pieces would cover the range of vitrification of commercial white-ware bodies, from the high porosity of wall tile to the low porosity of vitreous china, one half of the pieces were biscuit burned at cone 4 and the other half at cone 11.

GLOSS BURN.

All pieces were gloss fired at cone 3. The glaze and frit used had the following molecular formula:

Molecular composition of glaze.

$$\left.\begin{array}{l}0.25\ \mathrm{PbO}\\ 0.15\ \mathrm{K_2O}\\ 0.05\ \mathrm{Na_2O}\\ 0.45\ \mathrm{CaO}\\ 0.10\ \mathrm{ZrO_2}\end{array}\right\} 0.25\ \mathrm{Al_2O_3} \left\{\begin{array}{l}2.0\ \mathrm{SiO_2}\\ 0.4\ \mathrm{B_2O_3}\end{array}\right.$$

Molecular composition of frit.

$$\left.\begin{array}{l}0.10\ \mathrm{Na_2O}\\ 0.90\ \mathrm{CaO}\end{array}\right\} 0.1\ \mathrm{Al_2O_3} \left\{\begin{array}{l}2.0\ \mathrm{SiO_2}\\ 0.4\ \mathrm{B_2O_3}\end{array}\right.$$

The glaze was applied in a thick coat to one half of the trials by dipping and in a thin coat to the other half by spraying.

RESULTS OF TESTS.

TEST PIECES BISCUIT-FIRED AT CONE 12 AND GLOSS-FIRED AT CONE 3.

No black specks were observed in bodies 5, 6, and 7, containing treated Georgia kaolin, whereas a number of specks appeared in bodies 2, 3, and 4, especially in body 4, which contained 35 per cent of washed Georgia kaolin. The treated Georgia kaolin bodies had a better color than the washed kaolin. Body 5 had just as good a color as body 1, which contained English china clay, while body 6 had a slight creamy tinge and body 7 a decided cream color, when compared with body 1.

TEST PIECES BISCUIT-FIRED AT CONE 4 AND GLOSS-FIRED AT CONE 3.

A few black specks showed in body 4, but the difference in color between the treated and washed kaolin bodies was only slight at this heat treatment. Bodies 5 and 6 had just as good color as body 1, but body 7 showed a very slight creamy color when compared with 1.

CONCLUSIONS AS TO RESULTS OF COLOR TESTS.

Glazed white-ware bodies containing the treated Georgia kaolin have a whiter color than those containing the washed Georgia kaolin.

Treated Georgia kaolin can be substituted for at least 40 per cent of the English china clay in vitreous china, and for all the English china clay in wall-tile bodies without affecting the color of the product.

TESTS IN MAKING VITREOUS CHINA.

These preliminary tests gave indications of the possible commercial uses of the treated Georgia clay and justified further tests of substituting it for English china clay. In order that the tests might be made under commercial conditions of white-ware manufacture, they were conducted at the plants of the Beaver Falls Art Tile Co., the Mayer China Co., and the Mosaic Tile Co., through the courtesy of these firms.

The commercial tests of the use of treated Georgia kaolin as a substitute for English china clay in vitreous china bodies were made at the Mayer China Co. plant at Beaver Falls, Pa. The main object of these tests was to determine whether a practicable vitreous china body composed entirely of American materials could be made.

PROCEDURE IN TESTS.

MATERIALS USED.

The materials used in this investigation, with the exception of the treated Georgia kaolin, were taken from the stock bins of the china company. The following materials were used:

Feldspar.—Eureka Flint & Spar Co.
Flint.—Ohio Silica Co.
Florida kaolin.—Edgar Plastic Kaolin Co.
English china clay.—Moore & Munger's M. G. R. No. 2.
Ball clay.—Moore & Munger's English ball clay.
Treated Georgia kaolin.—Georgia Kaolin Co.

COMPOSITION OF BODIES.

The percentage composition of each body used in the three series of tests comprising this investigation are given in Table 5 following:

TABLE 5.—*Composition and physical properties of bodies used in tests.*

COMPOSITION.

Item.	Series A, body No.—						Series B, body No.—				Series C, body No.—					
	20	21	22	23	24	25	26	27	28	29	29b	30	31	32	33	34
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
Feldspar	16.0	16.0	16.0	16.0	16.0	16.0	17.0	18.0	16.0	16.0	16.0	16.0	16.0	16.0	17.0	16.0
Flint	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5
Ball clay	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	7.5	3.5	......	......	......	......
Florida kaolin	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0	8.0
English china clay	29.0	24.2	20.3	18.4	14.5	18.4	16.0	15.0	14.5	13.5	13.5	17.5	19.5	17.5	13.5	19.5
Treated Georgia kaolin	......	5.8	8.7	10.6	14.5	......	12.0	12.0	13.5	13.5	13.5	13.5	15.0	17.0	20.0	15.0
Washed Georgia kaolin	......	......	......	......	......	10.6	......	......	......	......	......	......	......	......	......	......
Whiting	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	3.0	4.0	4.0	4.0	4.0	3.0	2.0	2.0
Magnesite	......	......	......	......	......	......	......	......	......	......	......	......	......	1.0	2.0	2.0

PHYSICAL PROPERTIES.

Item.	20	21	22	23	24	25	26	27	28	29	29b	30	31	32	33	34
Total shrinkage	11.46	12.00	12.30	12.54	12.78	12.74	12.60	12.96	12.64	13.34	12.70	12.06	11.34	13.60	13.40	12.84
Drying shrinkage	2.42	2.60	2.78	3.00	3.17	3.16	2.84	3.00	3.20	3.16	3.08	2.24	2.50	3.14	3.22	2.58
Burning shrinkage	9.06	9.40	9.52	9.54	9.61	9.58	9.76	9.96	9.44	10.18	9.72	9.82	8.84	10.46	10.18	10.26
Absorption	0.05	0.41	1.00	1.08	1.30	1.02	0.26	0.03	0.60	0.02	0.16	1.19	2.26	1.05	0.76	1.04
	Cm.	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*	*Cm.*
Translucency	0.65	......	......	......	......	......	0.61	0.64	0.61	0.68	0.40	0.45	0.46	0.51	0.53	0.54

PREPARATION OF BODIES.

The weight of each ingredient taken was determined from the percentage composition, figured on the basis of an 8,000-gram batch for each body. After weighing and checking the total weight by weighing the batch 1.2 grams of cobalt sulphate was added for a stain, and the bodies blunged in the ordinary wet way in porcelain-lined ball mills. The bodies in series A were ground for only five hours, but after the bodies of this series were biscuited it was found that the blunging was insufficient to give the desired uniform texture to each body; therefore the bodies of series B and C were blunged for 20 hours. After thorough grinding, the bodies were passed through a 120-mesh sieve and dried to a plastic condition in plaster molds lined with cheesecloth. As soon as the bodies were sufficiently dry they were wedged on a marble slab, and then each body was made up into 10 disks 10 centimeters in diameter for shrinkage and absorption tests, and twelve 5-inch plates for color comparisons. Twenty-four wedge-shaped pieces for translucency tests were also made from each body of series B and C. These pieces were allowed to dry for about 12 hours and then fettled.

BISCUIT BURN.

The finished pieces were placed in saggers in the ordinary commercial way, and placed in the second ring of the biscuit kiln. The pieces in series A and B were burned to cone 10, but those in series C were burned only to cone 9, owing to the lower temperature in the kiln in which the test pieces of this series were burned.

GLAZING.

The glaze and frit used had the following molecular formula:

Molecular composition of glaze.

$$\left.\begin{array}{l} 0.058\ K_2O \\ 0.207\ Na_2O \\ 0.530\ CaO \\ 0.205\ PbO \end{array}\right\} 0.275\ Al_2O_3 \left\{\begin{array}{l} 2.540\ SiO_2 \\ 0.414\ B_2O_3 \end{array}\right.$$

Molecular composition of frit.

$$\left.\begin{array}{l} 0.272\ Na_2O \\ 0.031\ K_2O \\ 0.697\ CaO \end{array}\right\} 0.272\ Al_2O_3 \left\{\begin{array}{l} 2.550\ SiO_2 \\ 0.544\ B_2O_3 \end{array}\right.$$

Eight kilograms of the above glaze was ground in the ordinary wet way for 60 hours in a porcelain-lined ball mill, passed through a 120-mesh sieve, and flocculated or thickened by the addition of 0.13 per cent $CaCl_2$, calculated on the basis of dry weight. The thickened glaze was then applied to the biscuited plates by dipping in the ordinary commercial way.

GLOSS BURN.

The glazed pieces were gloss fired at cone 1 in the gloss kilns of the company.

RESULTS OF TESTS.

The physical and pyrophysical properties of the different bodies shown in Table 5 were compared with those of body 20, which was represented to the writer by Mr. Ernest Mayer as being a practicable vitreous china body. The physical and pyrophysical properties studied were as follows: Working properties, drying shrinkage, burning shrinkage, absorption, transluscency, and color.

The working properties of the bodies were studied as regards the ease which the ware was formed in the jigger without flawing.

The drying shrinkage and total shrinkage were determined by measuring the dry and burned lengths of 5-centimeter marks placed on the 10-centimeter disks while in the green state. The drying and total-shrinkage data given in Table 5 are expressed in percentage of green length. The burning shrinkage was calculated by subtracting the percentage of drying shrinkage from the percentage of total shrinkage.

The absorption data were obtained by weighing the dry disks and then soaking them in distilled water for 48 hours and again weighing. The average absorptions given in Table 5 are expressed in percentages of dry weight.

The translucency was determined by measuring the thickest part of the wedge-shaped piece through which light would pass. The average translucencies are given in centimeters in Table 5.

SERIES A TESTS.

The purpose of the tests in series A was to study the direct substitution of treated Georgia kaolin for English china clay in a practical vitreous china body

CONCLUSIONS FROM RESULTS OF SERIES A.

As the proportion of treated Georgia clay was increased the working properties of the body were improved, the drying and burning shrinkages and the absorption were increased. This decreased vitrification of the Georgia-kaolin bodies is not due to the difference in fineness of grain of the English and Georgia clays, as is commonly thought, for the Georgia kaolin is much finer grained than the English china clay, but is probably due to the low alkali content of the Georgia kaolins.

The colors of bodies 21, 22, and 23 were just as good as that of body 20, whereas body 24 showed a slight cream color, therefore only about 50 per cent of the English china clay in an ordinary vitreous china body can be replaced by this treated Georgia kaolin before the color of the glazed product is impaired.

SERIES B TESTS.

The tests in series B were designed to obtain a body of low absorption and still containing the maximum percentage of treated Georgia kaolin. For this purpose the feldspar content in bodies 26 and 27, and the whiting content in bodies 28 and 29 were increased.

CONCLUSIONS FROM RESULTS OF SERIES B.

The working properties of the four bodies of this series were very good. The burning shrinkage was increased, with resulting decreased absorption and increased translucency. The color of all the bodies of this series was just as good as that of body 20. Body 29 was a good vitreous china body of high translucency.

SERIES C TESTS.

The purpose of the tests of series C was to study the effect of substituting treated Georgia kaolin for ball clay, and magnesite for part of the whiting. Owing to the decreased biscuit heat the bodies of this series had an abnormally high absorption and low translucency, as may be seen by comparing the data for bodies 29 and 29b, Table 6. These two bodies had the same composition but were biscuit-burned with different heat treatments.

CONCLUSIONS FROM RESULTS OF SERIES C.

Substituting treated Georgia for ball clay decreased the drying and burning shrinkage and increased the absorption. The translucency was increased regardless of the decrease in vitrification.

Magnesite had a greater fluxing action than whiting, as is seen by comparing the absorption of body 31 with that of 34.

A direct substitution of Georgia clay for ball clay made the body too short, and therefore greater amounts of Georgia kaolin must be used, as compared with ball clay, to produce a practicable vitreous china body. Bodies 32 and 33, containing 17 per cent and 20 per cent of Georgia kaolin, respectively, had good plasticity, and worked well in the jigger, whereas bodies 30 and 31, which contained 13.5 and 15 per cent, were too short.

Bodies 31, 32, 33, and 34 of series C had a much better color than bodies 20 and 29 of series B, which contained ball clay. Bodies 33 and 34 had the best color of those in series C, which was evidently due to the substitution of magnesite for 50 per cent of the whiting.

GENERAL CONCLUSIONS AS TO PRACTICABILITY OF MAKING CHINA FROM GEORGIA CLAYS.

A commercial vitreous china body could not be obtained by the direct substitution of refined Georgia kaolin for English china clay, but the fluxes had to be increased to obtain the desired degree of vitrification.

The substitution of treated Georgia kaolin for all the ball clay and 50 per cent of the English china clay produced a body (No. 33) of good working qualities and as good a color as the whitest commercial ware and of a far better color than most vitreous chinaware on the market.

Body 33 contained 8 per cent of Florida kaolin, 13.5 per cent of English china clay, and 20 per cent of treated Georgia kaolin. As far as color is concerned, more than 20 per cent of treated Georgia kaolin can be used in bodies containing no ball clay, but the writer is of the opinion that if more than 20 per cent were added the temperature of the biscuit heat would have to be increased in order to obtain the desired vitrification, as an increase in percentage of fluxes over that in body 33 would reduce the toughness of the ware. As toughness is an important factor in high-grade vitreous china or hotel china, 20 per cent of treated Georgia kaolin is about the maximum proportion that can be used in this class of ware. In order to produce a vitreous china body containing only American clays it would be necessary in a body such as No. 33 to substitute for the 13.5 per cent of English china clay a domestic kaolin that is as easily vitrified as the English china clay and having as good a burning color.

MANUFACTURE OF WALL TILE.

The commercial tests of the substitution of treated Georgia clay for English china clay in wall tile were made at the factories of the Beaver Falls Art Tile Co., Beaver Falls, Pa., and the Mosaic Tile Co., Zanesville, Ohio. As the tests at the Beaver Falls plant were of a preliminary nature they will be discussed first.

PRELIMINARY TESTS.

The purpose of the preliminary tests at Beaver Falls was to determine the maximum proportion of Georgia clay that could be used in a commercial wall tile. The tests made at the Ohio State University showed that at least comparatively large proportions could be used in bodies of high absorption before the color was affected. What effect a large proportion would have on other pyrophysical properties of wall tile had next to be determined.

MATERIALS USED.

The materials used in this investigation were as follows:

English china clay.—Moore & Munger's M. W. M. No. 2.

Ball clay.—Kentucky No. 4.

Florida kaolin.—Edgar Plastic Kaolin Co.

Flint.—Pennsylvania Pulverizing Co.

Feldspar.—Pennsylvania Feldspar Co.

Treated Georgia kaolin.—Georgia Kaolin Co.

As these tests were to form the basis of larger commercial tests at the Mosaic Tile Co. plant, it was thought advisable to determine the fineness of grain of the flint and feldspar in order that the shrinkage and absorption could be controlled on changing to a different feldspar and flint. The mechanical analyses were made with a Schultze elutriation apparatus [a] having an overflow of 80° c. c. per minute, and are given in Table 6.

TABLE 6.—*Results of mechanical analyses of feldspar and flint.*

Residue.	Average diameter of grain.	Feldspar in residue.	Flint in residue.
Residue on sieves:	*Mm.*	*Per cent.*	*Per cent.*
On 120-mesh	0.186	9.46	0.60
On 150-mesh	0.157	0.90	0.42
On 200-mesh	0.110	7.50	1.56
On 260-mesh	0.0765	10.70	9.30
On 330-mesh	0.578	9.36	4.32
Residue from elutriator jars:			
In No. 1 jar	0.034	20.84	25.74
In No. 2 jar	0.0187	9.60	18.00
In No. 3 jar	0.010	10.80	19.06
Overflow from No. 3 jar	....	20.84	20.40

COMPOSITION OF BODIES.

The percentage composition of the bodies compounded from the above white-burning materials, and the physical properties of the bodies are shown in Table 7.

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[a] For description of apparatus, see Watts, A. S., Mining and treatment of feldspar and kaolin in the southern Appalachian region: Bull. 53, Bureau of Mines, 1913, p. 46.

TABLE 7.—*Composition and physical properties of bodies used in tests.*

COMPOSITION.

Item.	Series A, body No.—					Series B, body No.—			Series C, body No.—		
	10	11	12	13	14	15	16	17	18	19	20
	Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
Feldspar	15.0	15.0	15.0	15.0	15.0	13.0	12.0	11.0	12.0	12.0	12.0
Flint	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0
Florida kaolin	10.0	10.0	10.0	10.0	10.0	11.0	11.0	12.0	....	....	....
Kentucky ball clay	15.0	15.0	15.0	15.0	15.0	16.0	16.0	15.0	15.0	15.0	....
English china clay	20.0	15.0	10.0	5.0	....	....	5.0	....	33.0	....	....
Treated Georgia kaolin	....	5.0	10.0	15.0	20.0	20.0	16.0	20.0	....	33.0	48.0
Whiting	....	....	....	....	....	....	....	1.0	....	....	....
Magnesite	....	....	....	....	....	....	....	1.0	....	....	....

PHYSICAL PROPERTIES.

Item.	10	11	12	13	14	15	16	17	18	19	20
Shrinkage	9.70	9.57	9.37	9.15	8.51	6.39	6.10	5.76	6.74	4.59	3.41
Absorption	6.75	7.52	7.98	8.70	9.65	12.44	12.65	12.27	10.02	13.86	17.70
Shade (color) [a]	D	D	D	D	D	B	B	B	C	C	A

[a] Comparative, A is the whitest shade, D the darkest.

PREPARATION OF BODIES.

The weight of each ingredient to be taken was determined from the percentage composition, figured on the basis of a 10-kilogram body, and the total weight was checked by mixing the ingredients and weighing the batch. The bodies were blunged for five hours in a small blunger constructed from an ice-cream freezer; then the slip was passed through a 120-mesh sieve and filter-pressed in a small laboratory press. The press cakes were dried at 110° C. and then ground to pass a 20-mesh sieve. Thirteen per cent of water was added to the ground body, which was then thoroughly mixed by hand, and the moist clay reground in a small dust mill to pass a 30-mesh sieve. The finely ground clay was stored in stoneware jars and allowed to stand overnight.

MAKING THE TILE.

The bodies were pressed into standard 6 by 3 inch tile on an ordinary hand screw press. In order to obtain uniform pressing, all bodies were pressed by the same experienced pressman. Thirty tile were pressed from each body and divided into six sets of five tile each for distribution in different sections of the kiln.

BISCUIT BURN.

The tile were set in saggers in the ordinary manner. Each sagger contained five tile of each body composition, and were distributed in the kiln as follows: A, bottom of first ring, just above bog wall; B, top of first ring; C, bottom of second ring; D, top of second ring; E, bottom of third ring; G, top of third ring.

A plate containing Seger cones Nos. 9, 10, 11, and 12 was placed in each sagger. The heat treatment that the bodies in different sections of the kiln received was as follows:

Temperatures in different sections of kiln, as shown by Seger cones.

Kiln section.	Cone No.	Kiln section.	Cone No.
A	13	D	10
B	11	E	9
C	11	G	12

The burn was made in an ordinary down-draft Rivers kiln. The variation in temperature shown in the above table was due to a change in burners.

GLOSS BURN.

One tile of each body composition from each section of the biscuit kiln was glazed with the regular glaze used by the company. The glaze was applied by dipping.

VITRIFICATION.

A study of the vitrification was made by the determination of average absorptions and shrinkages. The absorption was determined by soaking the biscuited tiles in distilled water for 48 hours and calculating the percentage of water absorbed by each tile on the basis of the dry weight of the tile. The shrinkage was determined by measuring the length of the burned tile and expressing the difference between the green length (length of die) and the burned length in terms of percentage of green length. The different heat treatments which the test pieces in different sections of the kiln received had a decided effect on the absorption and shrinkage. This may be emphasized by the actual results of tests with body 18, which were as follows:

Results of absorption and shrinkage tests with body 18.

Tile in kiln section—	Average absorption.	Average shrinkage.
	Per cent.	*Per cent.*
A	7.94	7.43
B	10.19	6.90
C	9.75	6.88
D	10.66	6.55
E	12.30	5.83
G	9.29	6.88

The average shrinkage and absorption for each body have been given in Table 7.

DETERMINATION OF COLOR.

As there is no standard method of measuring the shade or color of a wall tile, the shade of the glazed trials was determined by comparison. The results are given in Table 7, A being the whitest shade and D the darkest.

RESULTS OF TESTS.

The purpose of the tests in series A, Table 7, was to study the effect of substituting treated Georgia kaolin for English china clay. The results show that as the content of Georgia kaolin was increased the vitrification was decreased, but no change in color was noted. All the bodies of this series had too high a shrinkage and too low an absorption for practical wall tile, as most commercial wall tile have a shrinkage of about 4.5 per cent and an absorption of 14 per cent.

The tests in series B, Table 7, were designed to decrease the vitrification shown by members 13 and 14 of series A. The data show that the shrinkage was reduced and the absorption increased, but not sufficiently to approximate those of commercial wall tile. An improvement in color, owing to decreased vitrification, was noted. Introducing 1 per cent of whiting and 1 per cent of magnesite did not improve the color of the tile.

The purpose of series C, Table 7, was to further reduce the vitrification by substituting treated Georgia clay for Florida kaolin and ball clay. The results speak for themselves. In bodies 18 and 19 33 per cent of treated Georgia kaolin was substituted for 33 per cent of English china clay without impairing the color of the glazed product. Another interesting fact was the improvement in color caused by substituting treated Georgia kaolin for ball clay in body 20, and in spite of the fact that the content of Georgia kaolin was 48 per cent of the body composition. The shrinkage and absorption of body 19, consisting of 12 per cent feldspar, 40 per cent flint, 15 per cent Kentucky ball clay, and 33 per cent treated Georgia kaolin, approximated that of commercial wall tile, and was therefore used as a basis for further investigations of wall-tile bodies.

LARGE-SCALE TESTS.

In the preliminary tests at the Beaver Falls plant the treated Georgia kaolin gave such promise of being an ideal clay for wall-tile manufacture that large commercial tests were thought advisable. These tests were carried on at the plant of the Mosaic Tile Co., Zanesville, Ohio.

MATERIALS.

The materials used in this investigation were as follows:

Flint.—Pennsylvania Pulverizing Co.

Feldspar.—Pennsylvania Pulverizing Co.

Florida kaolin.—Edgars Plastic Kaolin Co.
Ball clay.—Cooley ball clay.
Treated Georgia kaolin.—Georgia Kaolin Co.

COMPOSITION OF BODIES.

The percentage composition and physical properties of the bodies included in this investigation are given in Table 8.

TABLE 8.—*Composition and physical properties of bodies used in large-scale tests.*

COMPOSITION.

Item.	Series A, body No. —		Series B, body No. —									Series C, body No. —
	43	44	45	46	47	48	49	50	51	52	53	54
Flint............per cent..	40.0	40.0	40.0	40.0	40.0	40.0	40.0	40.0	37.5	35.0	35.0	35.0
Feldspar............do....	13.0	00.0	13.0	14.0	14.0	14.0	15.0	16.0	14.0	14.0	14.0	13.0
Cornwall stone.......do....	0.0	14.0	0.0	......	......	......	......	......	......	......	......	......
Florida kaolin.......do....	12.0	11.0	12.0	11.0	6.0	......	......	......	......	......	......	......
Ball clay............do....	0.0	0.0	0.0	......	5.0	11.0	5.0	......	13.5	16.0	10.0	14.0
Treated Georgia kaolin, per cent................	35.0	35.0	35.0	35.0	35.0	35.0	40.0	44.0	35.0	35.0	41.0	38.0

PHYSICAL PROPERTIES.

Item.	43	44	45	46	47	48	49	50	51	52	53	54
Shrinkage.......per cent..	3.00	3.00	5.28	5.44	5.92	6.40	6.56	7.20	6.88	7.84	7.52	4.67
Absorption..........do....	17.70	18.00	17.09	17.19	15.10	14.25	14.19	15.19	14.74	14.21	15.37	14.40
Breaking strength.pounds..	45.0	52.0	55.1	55.5	68.5	81.0	71.7	76.5	85.0	100.8	87.0	70.0
Relative shade (color).....	A	A	......	......	......	......	......	......	......	......	......	D

PREPARATION OF BODIES.

The bodies in series A and C, consisting of 3,000-pound batches, were prepared as follows: The total batch was blunged in a double blunger for one hour in the ordinary wet way and then the slip was passed through a 120-mesh vibrating screen into an agitator, from which it was pumped into the filter presses. The filter-press cakes were dried in a tunnel drier heated by the waste heat from cooling kilns and then crushed into small pieces in a set of rolls. About 11 per cent of water by weight was added to the crushed body and after standing for several hours it was finely ground in an ordinary dust mill of the fan type. The finely ground clay was collected in a large storage bin; from this bin it was taken to the presses.

The batch weight of the bodies in series B was only 8,000 grams, and they were prepared as follows: (1) The ingredients were weighed out and the total weight checked by weighing the batch. The clay was (2) blunged in porcelain-lined ball mills for 1 hour, (3) sieved through a 120-mesh screen, and (4) the slip concentrated in plaster absorption bowls lined with muslin, and dried completely in a drier. The dried clay was (5) crushed to pass a 20-mesh screen, (6) tempered by the addition of 11 per cent of water by weight, and the tempered clay was (7) ground to pass a 20-mesh screen, and (8) allowed to stand for several hours in a covered jar.

MAKING THE TILE.

The bodies were pressed into 3 by 6 inch and 6 by 6 inch tile on the power presses of the tile company.

BURNING.

The tile after being allowed to dry for about four days were set in saggers and biscuit burned at cone 11. The biscuited tile were then glazed with the regular glaze used by the company and gloss burned at cone 02.

RESULTS OF TESTS.

The absorption, shrinkage, and color data given in Table 8 were obtained in the same manner as in the preliminary tests. The breaking strengths given in this table were obtained by determining the weight that would break the biscuited tile with one end gripped between two steel plates. The weight was suspended 2½ inches from the point of support.

The object of the tests in series A, Table 8, was to compare the effects of feldspar and Cornwall stone in wall-tile bodies. The data in the table show that it takes 1 per cent more Cornwall stone to produce the same vitrification, but the stone increases the strength of the body about 15 per cent. The bodies of this series had too high an absorption, too low a shrinkage, and too low a breaking strength for good commercial wall tile.

The purpose of the tests in series B, Table 8, was to obtain a body of higher breaking strength. By comparing body 45 of this series with body 43 of series A it is seen that grinding a body in a ball mill, instead of blunging it, increased the shrinkage but had little effect on the absorption. The breaking strength was greatly increased by substituting ball clay for Florida kaolin, and by increasing the clay content at the expense of the flint a further increase in strength was obtained, reaching a maximum of 100.8 pounds in body 53. As none of the bodies of this series was gloss fired no relative shade or color is given for these bodies.

A large commercial test was made on a body of the composition of No. 54, in series C of Table 8. This body proved to be a good commercial wall-tile body. The low breaking strength, 70 pounds, is due to the fact that the finely ground clay was too dry when pressed. The color of the glazed tile was as good as that of some of the leading wall tile on the market.

APPLICABILITY OF REFINED GEORGIA KAOLIN IN WALL-TILE BODIES.

Treated Georgia kaolin can be substituted for all the English china content and a part of the ball-clay content in clay bodies without impairing the color of the burned product.

Cornwall stone makes a stronger body than feldspar, but a feldspar wall-tile body can be made that is as strong as any commercial wall tile on the market, provided that the total clay content is at least 52 per cent, of which 10 per cent is ball clay and the other 42 per cent is treated Georgia kaolin.

ACKNOWLEDGMENT.

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DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

THE FUSIBILITY OF COAL ASH AND THE DETERMINATION OF THE SOFTENING TEMPERATURE

BY

ARNO C. FIELDNER, ALBERT E. HALL
AND
ALEXANDER L. FEILD

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THE FUSIBILITY OF COAL ASH AND THE DETERMINATION OF THE SOFTENING TEMPERATURE.

By Arno C. Fieldner, Albert E. Hall, and Alexander L. Feild.

INTRODUCTION.

PURPOSE OF INVESTIGATION.

As a safeguard against excessive clinker troubles, specifications for the purchase of coal can be drawn to include the "softening" or "fusing" temperature of the ash. The value of such information has been recognized by the Bureau of Mines and has induced the bureau to investigate laboratory methods of determining the fusibility of coal ash, and the bearing of the results on clinker formation in fuel beds. The bureau realized that the fusibility of a mixture of oxides and silicates such as is comprised in coal ash varies according to the conditions under which tests are made, and that in the absence of any generally accepted method, no agreement is to be expected among tests made in different laboratories. Indeed, Marks[a] has recently called attention to differences as great as 390° C. that were obtained with the same sample of ash by two different laboratories. Therefore if a fusion-temperature clause is to have any place in specifications for coal, a standard method of testing should be adopted in order to insure comparable results by different laboratories and to obtain the softening or fusing temperature of the ash under conditions similar to those of a fuel bed. After a consistent method has been devised for obtaining comparable results at different laboratories, there will remain the correlation of these results with those from the burning of coal in furnaces before the value of fusibility tests in coal specifications can be finally determined.

SCOPE OF INVESTIGATION.

The present publication reviews the literature on the subject and gives in detail the effect of various oxidizing, reducing, and neutral atmospheres such as are found in various parts of the fuel bed on the softening temperature of ash when molded in the form of Seger cones. As a result of this study a practical method of determining fusibility has been developed whereby the ash is caused to soften and form slags in which the iron exists in approximately the same state of oxidation as the iron in fuel-bed clinkers.

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[a] Marks, L. S., The clinkering of coal: Power, vol. 40, 1914, pp. 932–934.

Although it is believed that this method will indicate the probable clinkering characteristics of a coal better than any fusion tests heretofore described, the bureau does not recommend the general use of this method in coal specifications until it is justified by actual trial in furnace tests of different coals. As no fusibility test of a well-mixed sample of the average ash of a coal takes into account the physical distribution of the impurities in the coal as burned, much additional investigation is needed to establish the exact relation of these laboratory fusibility tests to clinker formation. In order to correlate this proposed test of minimum softening temperature with clinker formation, the bureau is now making clinkering tests in a specially designed experimental furnace, in which different coals having ashes that differ in fusibility are burned under similar and known conditions of combustion. The results of this investigation will be published in a subsequent report. In the meantime it is hoped that other investigators will obtain similar correlation data.

HISTORICAL REVIEW.

EUGENE PROST, 1895 AND 1897.

Prost[a] analyzed the ash and determined the temperature of fusion of 23 different types of Belgian coals. The fusion tests were made by placing 3 grams of finely pulverized ash in a small porcelain crucible and then heating it for definite periods of time at four different temperatures, namely, 1,100°, 1,250°, 1,400°, and 1,500° C. The temperatures were controlled by using gold and platinum alloys of known melting points. After each period of heating the ash was examined. The progress of fusion was described as "barely fritted," "fritted," "strongly fritted," "incipient fusion," or "fused."

To obtain the different temperatures Prost used three different furnaces, as follows: (1) A Perrot gas furnace which gave a maximum temperature of 1,100° C.; (2) a coke furnace which gave a maximum temperature of 1,250° or 1,400° C., according to the method of operation; (3) a small blast furnace of fire brick which gave a maximum temperature of 1,500° C.

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[a] Prost, Eugene, Récherches sur les relations existant entre le degré de fusibilité et la composition des cendres de houille: Rev. Univ. Min., t. 31, 1895, pp. 87–98; Monit. Sci., t. 46, 1895, pp. 560–565; Sur les relations entre la composition et la fusibilité des cendres de houille: Bull. Assoc. Belge chim., t. 11, 1897, pp. 119–126; The fusibility and composition of coal cinders: Coll. Guard., vol. 70, 1895, p. 796; The relation between composition and fusibility in coal ash: Coll. Guard., vol. 74, 1897, p. 602; Fusibility of coal ash determined by its composition: Coll. Guard., vol. 75, 1898, p. 473.

In furnaces 2 and 3 the crucibles containing the ash samples and the alloy-temperature indicators were placed in a fire-clay crucible with a wall 1 cm. thick. Coke was packed around this crucible. Prost calculated from the chemical analysis the refractory quotient:

$$Q=\frac{\dfrac{\text{O in } Al_2O_3}{\text{O in } CaO+\text{O in } Fe_2O_3+\text{O in } MgO}}{\dfrac{\text{O in } SiO_2}{\text{O in } Al_2O_3}}$$

for 23 Belgian coal ashes; he found that in general the fusing temperature increased with the value of Q. There were, however, several exceptions to the calculated results, especially for ash of high iron content. Prost's method of determining relative fusibility was merely approximate; for the conditions of reduction or oxidation undoubtedly varied in his different furnaces. Also his estimate of the degree of fusion on mere observation of the appearance of the melt in a crucible would necessarily be rather indefinite, for judgment of different observers would vary.

HENRY LE CHATELIER, 1902.

Le Chatelier[a] used the Seger-cone method for determining the relative fusibility of different coal ashes. He burned the coal in a muffle at a low temperature, ground the ash fine, and then, using starch as binder, molded the ash into triangular pyramids 50 mm. (2 inches) high and 15 mm. ($\frac{9}{16}$ inch) wide at the base. These pyramids were ignited to destroy the organic binder, placed in a gas furnace, with a suitable number of Seger pyrometric cones, and slowly heated until they collapsed. The temperature at which the vertex of a pyramid became half the original height of the cone was taken as the softening temperature. Le Chatelier gives no data on the possibility of reducing gases from the furnace atmosphere acting on the iron oxide of the ash.

J. W. COBB, 1904.

Cobb[b] ground coal ash to pass a 100-mesh sieve or to an impalpable powder, and, using dextrin solution for binder, made the ash into four-sided pyramids 3 inches (76 mm.) high and 1 inch (25 mm.) square at the base. Each pyramid was mounted with one face vertical, and placed in a circular, down-draft, regenerative furnace, heated by eight burners on the periphery.

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[a] Le Chatelier, Henry, Ètude sur la fusibilité des cendre des combustibles: Bull. Soc. L'Ind. Nat., t. 102, 1902, pp. 223–229. Abstract (Fusibility of the ashes of fuels) in Jour. Soc. Chem. Ind., vol. 21, 1902, p. 459.

[b] Cobb, J. W., Coal ash: Jour. Soc. Chem. Ind., vol. 23, 1904, pp. 11–13.

Temperature measurements were taken with Seger pyrometric cones and with a thermocouple. The temperature of the furnace was raised slowly until the tips of the cones started to bend. The rate of heating was then checked to almost zero to allow the cones to bend slowly through a small range of temperature. Cobb gives no exact figures for rate of heating, but calls attention to the necessity of a uniform rate to insure comparable results, and also states that an oxidizing atmosphere must be maintained in the space surrounding the cones.

E. G. BAILEY AND W. B. CALKINS, 1910 AND 1911.

In 1910 Bailey and Calkins[a] published a paper giving experimental data on the relation between the fusing temperature of ash and the formation of clinker. Tests were made in two hand-fired furnaces with several coals having ash of different fusing temperatures. All clinkers larger than about 1 inch were picked out of the refuse and the ratio of clinkers to the total ash in the coal burned was determined. The results, plotted in curves, showed a rapid increase in the ratio of ash fused to clinkers as the fusing temperature fell below 1,371° C. (2,500° F.). The clinker made from a lower fusing ash obstructed more grate area than the same weight of clinker formed from an ash of higher fusing temperature.

In a subsequent paper Bailey[b] describes two 24-hour boiler tests with two coals A and B. The fusing temperatures of the ashes of these coals were 1,510° C. (2,750° F.) and 1,310° C. (2,390° F.). On the basis of the total ash in the coal, coal A formed 20.2 per cent and coal B 53.3 per cent of clinker. Also, the clinker from coal B was much more troublesome in that the clinkers were thinner, thus obstructing about one-third more grate area than the same weight of clinker from coal A. In several boiler tests of coals, made by Prof. E. A. Hitchcock at the Ohio State University, two coals, C and D, having ash that fused at 1,260° C. (2,300° F.) and at 1,450° C. (2,640° F.), were burned on a plain grate of 50 per cent air space, under a Dutch oven. The ash with the lower fusing temperature had a greater proportion fused to clinker than did the ash with the higher fusing temperature; also the clinker from coal C was thin and vitreous, thus offering more obstruction to the passage of air through the fuel bed than the fragile and porous clinker resulting from coal D.

---

[a] Bailey, E. G., and Calkins, W. B., Fusing temperatures of coal ash: Power, vol. 32, 1910, pp. 1978–1979. Fuel Testing Co. Bull. 5, 1911, 15 pp.

[b] Bailey, E. G., The fusing temperature of coal ash: Power, vol. 34, 1911, pp. 802–806; 1910, pp. 1978–1979.

Coals E and F, having ash that fused at 1,210° C. (2,210° F.) and 1,400° C. (2,550° F.), were burned on shaking grates under a firebrick arch of a special horizontal return tubular boiler setting. These two coals did not show the difference in clinker formation that was expected from the difference in the fusing temperatures of the ashes. This result is ascribed to a probable difference in fuel-bed conditions and a difference in the burning characteristics of the two coals. However, the lower fusing ash from coal E formed a thinner clinker than coal F, obstructing approximately 0.21 square foot of grate area per pound, as compared to 0.13 square foot for the clinker from coal F.

Bailey concludes that the "main factor in the formation of clinker is the difference between the fusing temperature of the ash and the temperature to which it is subjected. Any factor, such as excess air, rate of combustion, and thickness of fuel bed which will affect the temperature may cause a corresponding change in the amount and nature of the clinker from a given coal."

L. S. MARKS, 1910.

Marks[a] used the Seger-cone method for determining the effect on the fusibility of ash of adding various proportions of Al_2O_3, CaO, Fe_2O_3, FeS_2, Fe, CaS_2, and $CaSO_4$. The finely ground mixtures were molded into pyramids similar in shape and size to standard pyrometric cones, then mounted in a vertical position and slowly heated in a gas muffle. The temperatures were measured with a thermocouple. Sketches were made of the cones as they melted, and the corresponding temperatures were noted. The viscosity of the softening material was judged from the rapidity and manner of deformation of the cone. Certain mixtures melted to a fluid of the viscosity of water, others formed a spongy, viscous mass. Marks obtained the following results on adding various substances to the ash of a nonclinkering West Virginia coal, which had a softening temperature of 1,400° C. (2,550° F.):

Results of tests by Marks with the ash of a nonclinkering West Virginia coal in a gas muffle.

Substance added.	Effect on softening temperature of ash of adding—			Fluidity of melt.
	25 per cent.	50 per cent.	75 per cent.	
Al_2O_3	Raised	Raised	Raised	Very fluid.
CaO	Lowered considerably		do	
Fe_2O_3	Lowered slightly	Lowered slightly	Lowered slightly	
FeS_2	Lowered considerably	Lowered considerably	Lowered considerably	Viscous.
Fe	do	Lowered somewhat	Lowered slightly	
CaS_2	Lowered	Lowered considerably	Lowered considerably	Very fluid.
$CaSO_4$	do	do	do	Do.

---

[a] Marks, L. S., The clinkering of coal: results of tests for effect of various constituents in the coal: Eng. News, vol. 64, 1910, pp. 623–626.

Similar clinkering tests were then made in a specially designed furnace, analogous results being obtained. Substances that lowered the softening temperature of the ash tended to increase the clinker formation. The addition of 3 per cent of pyrite (FeS_2) to a nonclinkering West Virginia coal containing 6 per cent ash produced a hard, heavy clinker, which had a strong tendency to stick to the grate bars, thus choking the draft. Adding 2.2 per cent of ferrous sulphide (FeS) caused a similar degree of clinker formation. The addition of 4.4 per cent of ferrous sulphide made a very fusible and extremely troublesome clinker, which flowed over and "froze" to the grate bars, seriously checking the draft. Addition of iron oxide with an iron content equivalent to that in the 3 per cent pyrite produced a somewhat less troublesome clinker than did the pyrite.

The tests seemed to indicate that, other things being equal, the sulphides of iron are more troublesome than the oxides.

E. J. CONSTAM, 1913.

Constam,[a] in 1913, reported the "melting temperatures" of the ash from 200 different samples of coal and coke from various European mines. His criterion of melting temperature was the deformation or softening point of the ash when molded in the form of a small pyramid. The pyramids were heated in a carbon-tube electric furnace having a magnesia tube closed at one end in which the ash pyramid was placed. The temperatures of incipient softening, intumescing, and complete fusion were measured with a Holborn-Kurlbaum optical pyrometer. Constam gives no information on the rate of heating, exact size and shape of the test piece, or the character of the atmosphere in which it was heated, in his published report. At the temperatures necessary to fuse coal ash the magnesia tube would not protect the ash from the reducing action of the carbon monoxide evolved from the heating element, unless a vigorous current of air was circulated through the interior of the tube. In the absence of complete information as to whether oxidizing or reducing atmospheres were used in the fusion tests, Constam's results can not be properly classified, but they gave excellent agreement of the softening temperatures reported for different samples of coal from the same source, and for samples of ash from a given coal and from its resulting coke.

As a result of his investigations, Constam concludes that (1) the proportion of ash does not influence the softening temperature;[b] (2) the softening temperature is characteristic of the seam; (3) the softening temperature is not affected by coking the coal or by the temperature of coking; (4) the softening temperatures of different ashes ranged from 1,150° to 1,700° C.

---

[a] Constam, E. J., Ueber Schmelztemperaturen von Kohlenaschen: Ztschr. Ver. Gas- und Wasserfachmänner in Oesterreich-Ungarn, Oct. 15, 1913. Abstract in Jour. Gas Lighting, vol. 124, 1913, p. 522.

[b] This conclusion does not hold when the ash contains an excess of a single constituent, as pyrite or slate.

Regarding various fuels, he recommends temperatures above which the ash of each fuel should fuse, as follows: Coke for a central heating plant, 1,300° C.; anthracite for the same purpose, 1,400° C.; coal for use under boilers, 1,400° C.; coal for locomotives, 1,500° C.; and coal for gas producers, about 1,500° C., or below 1,200° C.

L. S. MARKS, 1914.

Following the publication of Bailey's papers in 1910 and 1911, the growing demand for the fusibility test in connection with the purchase of coal under specifications led to the development of such tests in different laboratories. Although most of these tests depended on some modification of the well-known Seger-cone method used by Le Chatelier and by Cobb, highly discordant results were obtained by different laboratories with samples of the same ash. In 1914, Marks[a] called attention to differences as high as 390° C. in the fusing temperatures reported by different testing laboratories for samples of the same ash. An investigation of the causes of these variations led to the conclusion that the most important factor was the atmosphere surrounding the ash. Using the Seger-cone method, Marks determined the softening temperatures of 11 samples of ash in a Hoskins carbon resistance furnace and in a Meker muffle furnace; for each sample the Hoskins furnace gave the higher softening points, the difference ranging from 120° to 255° C. This variation in fusibility was attributed to the Hoskins furnace containing a reducing atmosphere of carbon monoxide and the Meker furnace having an oxidizing atmosphere from the air circulating through the muffle. Marks considers it essential that fusion tests be made in an oxidizing atmosphere.[b]

As less important causes for variations in the results of Seger-cone tests, Marks enumerates the following: (*a*) Size of cone; (*b*) position of cone; (*c*) nature of binder; (*d*) rate of heating; (*e*) location of cone in furnace; (*f*) material on which the cone is supported; (*g*) unburned carbon in the ash.

The use of a 10 per cent dextrin solution for binder and slight variations in the size of the cone had no appreciable effect on the results.

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[a] Marks, L. S., The clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 205–208; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 801–834.

[b] Combustion tests made for the Bureau of Mines by Henry Kreisinger, engineer, have shown that at ordinary rates of combustion almost all the oxygen in fuel-bed gases has disappeared at 3 inches above the grate bars, the gases at this point containing approximately 15 per cent CO_2 and 14 per cent CO, the remainder being nitrogen and other combustible gases. It is therefore highly probable that clinker forms in a slightly reducing atmosphere, and that fusion tests of coal ash in air do not parallel furnace conditions.

The deformation temperatures of 11 different samples of ash tested in horizontally mounted[a] cones were from 5° to 50° C. lower than when similar samples were tested in vertically mounted cones. The average difference was 21° C. The smaller differences were observed with samples showing the greatest fluidity. Cones mounted horizontally were more sensitive to heat than those mounted in the usual manner, and gave indication that could be duplicated more closely.

Comparative tests of the effect of rate of heating with two samples of ash showed deformation points 40° and 35° C. lower at 6° increase per minute than at 2° per minute. This difference was ascribed to the increased lag of the thermocouple and its protecting porcelain tube at the higher rate of heating.[b]

Unburned carbon in the ash tended to increase the softening temperature.

Marks finally adopted as the most satisfactory modification of the Seger-cone method the molding of the ash into a cone 11 by 52 mm. ($\frac{7}{16}$ inch by 2 inches), placed in a horizontal position on a quartz plate, with the apex projecting over the side of the support. The quartz plate was blocked up near the center of the muffle of a No. 29 Meker furnace. A stream of air provided an oxidizing atmosphere. A thermocouple, protected with a porcelain tube, was inserted through a hole in the back of the furnace, the cone being placed as near it as possible. The furnace was heated rapidly to within about 200° C. of the expected fusing temperature; the rate of temperature increase was then reduced to 2° C. per minute. The temperature of initial and of final bending (apex of the cone pointing vertically downward) was noted, and also the final appearance of the cone—whether it showed a fluid constituent that ran down and formed a knob on the vertex of the cone. This seemed to be somewhat characteristic of clinkering coals.

The results of tests in which the method of determining fusibility, described above was used, were compared with the clinkering results observed when 10 different coals were burned under normal power-house conditions. Marks, concludes that "a general relation was shown between the two, but not definite enough to be reliable. For the particular boiler plant investigated, it would seem that an ash with a fusing temperature (final bending) below 1,400° C. will probably give trouble if the ash cone shows a fusible constituent; whereas it will not give trouble with a fusing temperature above 1,380° C. if the ash is viscous. This conclusion would require further investigation with many other coals before it could be accepted even for this particular plant; naturally it can not be applied to plants with different operating conditions."

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[a] As recommended by J. P. Sparrow of the New York Edison Co.

[b] Higher rates of heating should give higher softening temperatures owing to the time lag in the slag-forming reactions. This relation has been shown in experiments with Seger pyrometric cones by numerous investigators. See Zimmer, W. H., Practical experience with pyrometers: Trans. Am. Cer. Soc., vol. 1, 1899, pp. 23–38; Geijsbeck, S., The fusing points of Seger cones expressed in degrees: Trans. Am. Cer. Soc., vol. 6, 1904, pp. 94–99; Geijsbeck, S., The melting points of pyrometric cones under various conditions: Trans. Am. Cer. Soc., vol. 14, 1912, p. 849; Hoffman, G., Prüfung der Segerkegel; Sprechsaal, Bd. 44, 1911, pp. 143–145; Reike, R., Die Schmelzpunkte der Segerkegel 022–15; Sprechsaal, Bd. 44, 1911, pp. 726–728; Brown, G. H., and Murray. G. A., The function of time in the vitrification of clays: U. S. Bur. Standards Tech. Paper 17, 1913, p. 23.

F. C. HUBLEY, 1914.

Various commercial laboratories that have developed individual modifications of the fusibility test seem to find a somewhat definite relation between the indications of their particular laboratory test and the clinkering characteristics of the fuel used in the power plants under their observation. Hubley,[a] after two years' experience with the Seger-cone method, considers it of value in predetermining the clinkering properties of a coal, and states that in most instances the clinkering of a coal in the boiler furnace agreed closely with the results of the laboratory tests.

Hubley used cones 2 inches high and 1 inch in diameter at the base. He ground the ash to pass a 60-mesh sieve; mixed it with 10 per cent by weight of dry dextrin and water enough to make a plastic mass, which was pressed in wooden molds, dried, and heated in a vertical position in a cylindrical Fletcher crucible furnace. Illuminating gas was used for heating to temperatures up to 2,700° F., and Blau gas for temperatures to 3,000° F. The temperatures were measured with a thermocouple, having the bare hot junction close to the cone. The temperature was gradually increased until signs of fusing appeared, as when the sides began to lose their sharp outlines, or when the body of the cone began to swell. The furnace was held at this temperature for three minutes, and then allowed to cool. Other cones from the same ash sample were in turn tested, the final temperatures for the lot being regulated to give a series at 25° or 50° F. intervals on either side of the temperature obtained in the preliminary test. Enough cones were tested to indicate a series from a temperature of no deformation to one at which the ash flowed readily. From this series the progress of fusion at various temperatures was determined.

This method gave satisfactory indication of probable clinkering characteristics, but proved tedious and somewhat lacking in definiteness, as the judgment of the operator entered too largely into the results.

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[a] Hubley, F. C., Bituminous coals; predetermination of their clinkering action by laboratory tests: Proc. Eng. Club of Philadelphia, vol. 32, January, 1915, pp. 35–85; abstracted in Power, vol. 40, 1914, p. 796; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 208–211; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 815–824.

To remedy these objections, Hubley devised an apparatus, which he called a "fusiometer," for indicating mechanically the progressive softening of a pellet of ash. This apparatus consists of an iron frame carrying a small gas-fired furnace, into which project two carbon rods, one-half inch in diameter on a common vertical axis. Between the carbon rods is placed the ash sample, compressed without binder, under a pressure of 30,000 pounds per square inch, into the form of a cylindrical pellet five-eighths of an inch in diameter and one-half inch high. The upper carbon rod, free to move vertically between suitable guides, is weighted to exert a pressure on the ash pellet of 1.5 pounds per square inch. A silk cord connecting the upper end of the rod and a counterweight runs over a pivoted pulley to which is attached a pointer magnifying twelve times on a scale any expansion or contraction of the pellet. The temperature of the furnace is measured with a thermocouple. Temperatures up to 2,700° F. may be attained with illuminating gas, and above 3,000° F. with Blau gas.

The method of making a test is described by Hubley as follows:

> The ash pellet is placed in position between the carbons and adjusted so that the pointer is midway between 0 and 1.00 on the scale. The furnace is then gradually heated at the rate of 50° to 100° F. per minute, simultaneous temperature and scale readings being made at ½-minute intervals. As the heat is increased, negative movement of the pointer will indicate carbon expansion up to a temperature where first softening of the ash pellet is indicated by a positive movement of the pointer. The experiment is continued till final collapse of the pellet is indicated on the scale. These results, if plotted, the temperatures as abscissæs, and the pointer movement as ordinates, produce a curve, an ordinate of which at any point is a measure of the relative rate of softening of the ash pellet at that temperature.
>
> The length of the softening range, and its position on the temperature scale in relation to the working temperature range of a boiler fire, as well as the increasing rate of softening, whether gradual throughout the range (indicating high viscosity), or very slight for most of the range, followed by a sudden collapse, are the factors in the fusiometer results which must be considered in predicting the probable clinkering action of a coal in a boiler fire. A long range ash with a slow but steady increase in softness, apparently indicated by the gently sloping curve, is productive of the close gummy clinker of high viscosity. This type, provided the softening range coincides approximately with the working temperature range of the boiler, produces the greatest losses from clinker formation in a boiler fire. This refers more particulaly to stokers in which the fires are cleaned at regular intervals by a dropping of the back grates.
>
> The other type of ash fusion is a short-range, spongy, porous formation of low viscosity, which, if it does not occur too low on the temperature scale, can be handled with ease in the boiler fires. From a comparison of boiler-house results with a large number of tests made, the fusiometer curve for this type of fusion appears to be distinguished by a most decided downward dip in the curve just prior to final softening.
>
> The term "fusing point" is misleading and indeterminate, while "fusing range" of a substance can be determined with exactitude. For purposes of

relative comparison, however, an arbitrary point in the fusing range may be selected and called the "fusing point" of the ash. For this point in the case of the fusiometer results, the author suggests the temperature at which the pellet has collapsed, owing to softening, to one-half of its original height.

Hubley shows a number of fusiometer curves illustrating the two types of fusion and discusses them together with the boiler-house reports in regard to the nature and extent of clinker formed under actual operating conditions. He states that usually he found these curves of much use in indicating the clinkering characteristics of the coals represented, and suggests the following specification:

> The pellet shall not have collapsed due to softening to more than one-half of the original height at a temperature, T, and in addition shall not show a collapse of more than one-eighth of the original height due to softening at a temperature of 300° F. under temperature T.
>
> The first part of this specification will limit the final fusion, while the second part is intended to control viscosity and avoid a sloping curve. The fixing of temperature T must vary with plant conditions. It is obvious that the buying field for steam coal will be restricted or broadened according as the temperature T is raised or lowered, so that the fixing of this temperature must be governed by the value which a plant management places on a loss of boiler capacity, and risk of steam failure against the saving made in purchasing the cheaper fuel.

The excellent agreement Hubley obtained on duplicate samples of the same ash does not prove that the method will give correspondingly close agreement in different laboratories, as variations in the slagging reactions of the ash will affect results obtained by this method as well as those obtained by the Seger-cone method. The principal factors would be the nature of the atmosphere—whether oxidizing or reducing—and the rate of heating. If a gas furnace is used, the nature of the atmosphere will vary with different adjustments of gas to air. The effect of different atmospheres on the softening of cones is described elsewhere. Undoubtedly their effect will be similar in the fusiometer furnace.

O. W. PALMENBERG, 1915.

Palmenberg[a] states that the Seger-cone method, modified as described below, is used in his laboratory, and has proved satisfactory in furnishing laboratory results that are comparable with the results obtained in practice.

> The coal ash which has been burned free of carbon and heated to insure oxidation of the iron is made into a round, thin cone 1½ to 2 inches high, weighing about 1 gram or less. This cone is placed upon a thin piece of Battersea fire clay in a vertical position and set in the middle of the muffle of a Meker

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[a] Palmenberg, O. W., Discussion of paper on the clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 211–212; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 824–828.

gas furnace. After heating the furnace, the blast is applied rapidly at first until the temperature is nearly that at which the cone begins to bend. Then the heat is regulated carefully and temperature readings with an optical pyrometer made every five minutes or oftener, depending upon the rate of heating; care must be taken to have the heating carried on slowly. As soon as the cone shows signs of bending, the temperature is noted and the operation continued until the cone has completely bent over and touched the plate, the temperature at which the cone touches the plate being called the fusing temperature.

Palmenberg prefers using long, thin cones, which in his experience are not subject to "separation of the more fusible constituents," an objection raised by Marks against the vertical-cone method. In regard to the effect of unburned carbon in the ash, Palmenberg states that this depends on the conditions of the tests and the nature of the ash; he found that as much as 10 per cent carbon had no effect, as seemingly it all burned off before the fusion took place.

E. B. RICKETTS, 1915.

Ricketts[a] describes the Seger-cone method as used in the laboratory of the New York Edison Co. as follows:

About 2 pounds of coal is coked in a crucible in a gas furnace for about an hour, after which the lump of coke is broken up and a stream of compressed air is fed in near the bottom of the crucible for from two to three hours until the coke is all reduced to ash. When burning the coke down, care should be taken to keep the temperature below 1,500° F. (820° C.), so as to prevent premature fusion. The ash is then tested, a small amount at a time, with oxygen to make sure that all the carbon is consumed. It is of vital importance that no carbon be left in the ash as a small trace may cause results several hundred degrees too high. The carbon-free ash is moistened with a little water and molded in a paper cone 2½ inches long and ½-inch base. The cone is dried for five or six hours and then fused in a muffle furnace without removing the paper in which it was molded. The cone is mounted horizontally so that it overhangs the point of support about 1½ inches. The point of fusion is taken when the cone reaches a 45° position and is read on a Wanner optical pyrometer.

The furnace consists of a gas melting furnace of 8 inches inside diameter. It is heated by horizontal blast burners which direct the flame around and over a covered Hessian crucible containing the ash cone. Observation of the interior of this crucible and of the cone is made through a suitably placed quartz tube which passes through the side of the melting furnace and the Hessian crucible.

Ricketts states further that with this method results of different tests of the same sample of coal corresponded within a few degrees and that the boiler tests with Babcock & Wilcox boilers fired with a Taylor stoker, showed a relation between efficiency of furnace and boiler and the fusion temperature of the ash. Thirteen coals were tested; the ash-fusion temperatures being between 2,225° F. (1,220°

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[a] Ricketts, E. B., Discussion of paper on the clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 213–214; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 830–834.

C.) and 2,850° F. (1,570° C.). Those coals whose ash fused below 2,375° F. (1,300° C.) gave a distinctly lower efficiency than those whose ash fused above 2,450° F. (1,345° C.). The clinkering effect of coals whose ash-fusion temperatures were between the two temperatures mentioned was doubtful. The results of ash-fusion tests of coal from several hundred cargoes, from some 25 different sources, were said to confirm consistently the results of the earlier tests.

COMPOSITION OF ASH AND DISTRIBUTION OF ASH-FORMING CONSTITUENTS IN COAL.

Ash is the incombustible residue left from the complete combustion of the carbon in coal. It is derived from the inorganic matter in the coal, and is composed largely of compounds of silica, alumina, iron, lime, together with smaller quantities of magnesia, titanium, alkali, sulphur, and phosphorus compounds. The silica, alumina, and titanic oxide are derived from sand, clay, shale, and slate, the iron oxide mainly from iron pyrite, and the lime and magnesia from their corresponding sulphates and carbonates.

As shown in the ash analyses given in Table 2 (p. 27), the chemical composition varies so widely that no typical composition can be given. Probably the analyses of most samples of coal ash will come between the following limits:

Typical limits of ash analyses.

Constituent.	Per cent.
SiO_2	40 to 60
Al_2O_3	20 to 35
Fe_2O_3	5 to 25
CaO	1 to 15
MgO	0.5 to 4
Na_2O+K_2O	1 to 4

The ash-forming constituents consist of (1) "inherent" or "intrinsic" impurities that are present in an intimate mixture with the coal substance, and are derived either from the original vegetable material or from external sources by sedimentation, precipitation, or aeolic action during the process of accumulation of coal-forming vegetation; (2) impurities formed either during or after the laying down of the coal bed, which occur in the form of partings, veins, and nodules of clay, shale, slate, pyrite and calcite; and (3) impurities that become mechanically mixed with the coal in the process of mining, such as fragments of roof and floor.

INTRINSIC ASH.

The ash-forming constituents comprising the intimately mixed impurities may be obtained by placing coarsely crushed coal in a solution of calcium or zinc chloride having a specific gravity of 1.35. The coal will float, and the heavier extraneous impurities will sink. In some coals the percentage of intrinsic ash is so low that the ash-forming constituents must have come almost wholly from the coal-forming vegetation. A noteworthy example of coal exceptionally low in ash is that in the Sewell bed of West Virginia; several mine samples taken from this bed near Beckley, in Raleigh County, showed on analysis 1.5 to 2 per cent ash. Ordinarily the intrinsic ash is between 2 and 8 per cent, although in many coals it may reach a fairly high percentage, especially in bony coals, which yield 25 to 40 per cent ash.

In the minute joints or cleavage planes of certain coals are thin flakes of calcite, gypsum, silica, or clay, and some coals, as certain Arkansas coals, have veinlets of pyrite. These impurities are not intimately mixed with the coal substance, but yet can not be separated from the coal by any commercial methods of preparing the coal for use. Microscopic crystals of pyrite are also found distributed to such an extent in some coals that washing produces no appreciable lowering of the sulphur content.

The "intrinsic" ash may be either more or less fusible than the total ash of the coal as mined, depending on the composition of the two kinds of ash. In general, the ash from a coal having a high intrinsic ash content, such as is found in bony coal, is less fusible than the ash from a coal having low intrinsic ash content, as the high ash content is due to an intimate admixture of siliceous or argillaceous silt, which increases the softening temperature. On the other hand, a coal of normally low intrinsic ash content, containing well distributed microscopic crystals of pyrite, or joints filled with flakes of calcite, gypsum, or pyrite, is likely to form troublesome clinkers when burned. Such a coal contains well mixed and finely divided essential constituents of an easily fusible slag.

EXTRANEOUS ASH.

The percentage of ash derived from extraneous impurities varies considerably, depending on the number and size of the partings in the seam, the possibility of separating these from the coal, and the care with which the coal is mined. Impurities of this character may be removed to some extent by suitable methods of washing, picking, or screening. The possibility of improving the quality of coal by washing may be determined by the laboratory float-and-sink test already described; also, by determining the fusibility of the ash of the raw coal, as compared with that of the float coal, the effect of better preparation on the fusibility of the ash may be determined.

Better preparation may either increase or decrease the ash fusibility, depending on the fusibility of the intrinsic ash and the nature of the extraneous impurities. Such extraneous impurities as slate, sandstone, or shale tend to raise the softening temperature; limestone and pyrite tend to lower the softening temperature, although limestone, if added in excess, will increase the softening temperature. Experience has shown that many of the best low-ash coals form very troublesome clinkers. A very fusible intrinsic ash would be expected to give excessive trouble in the fuel bed, as the various constituents are finely divided and intimately mixed in the coal, thus making a favorable condition for slag formation. It is doubtful whether large lumps of extraneous impurities have much influence on slagging in the furnace. Large pieces of slate are usually found unfused in the refuse.

NATURE OF FUSION OF ASH.

MELTING POINTS OF PRINCIPAL CONSTITUENTS OF ASH.

According to Day and Sosman,[a] "the melting point of a pure substance may be defined as the temperature at which the crystalline and liquid substance can remain side by side in equilibrium. Only the addition or withdrawal of a quantity of heat will cause the disappearance of one of the two forms in contact. Melting is therefore characterized by two concurrent changes, the appearance or disappearance of a particular crystal structure, and the appearance or disappearance of a quantity of heat. This definition applies to pure compounds. If the material is a mixture or a solid solution, it will have, not a melting point, but a melting interval, with (theoretically) definite temperature limits."

According to this definition, the principal constituents of coal ash, considered by themselves and in their crystalline form, have melting points well above the temperature at which fusion in the mixed ash is known to begin. Quartz (SiO_2) at temperatures above 1,600° C. changes rapidly into cristobalite, which melts at about 1,625° C.[b]

At this temperature the rate of melting is very slow; and the fused silica has such a high viscosity that it does not flow or change shape distinctly until a temperature of 1,750° C. is reached.[c]

Pure alumina (Al_2O_3) melts at 2,050° C.[d]

Kaolinite (Al_2O_3, $2SiO_2$, $2H_2O$), the crystalline form of the essential constituent of clay, is unstable at high temperatures and appears to decompose into aluminum silicate (Al_2O_3, SiO_2) and free silica.

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[a] Day, A. L., and Sosman, R. B., The melting points of minerals in the light of recent investigations on the gas thermometer: Am. Jour. Sci., vol. 31, 1911, p. 342.

[b] Fenner, C. N., The stability relations of the silica minerals: Am. Jour. Sci., vol. 36, 1913, pp. 331–384.

[c] Kanolt, C. W., Melting points of fire bricks: U. S. Bur. Standards Tech. Paper 10, 1912, p. 14.

[d] Kanolt, C. W., Melting points of some refractory oxides: U. S. Bur. Standards Bull., vol. 10, 1913, p. 15.

Sillimanite (Al_2O_3, SiO_2) melts at 1,816° C.[a] Kanolt[b] found the temperature of visible flowing of an English white kaolin to be 1,740° C. A sample of brownish-white German kaolin began flowing at 1,735° C. Lime (CaO) melts at 2,572° C.,[c] and magnesia (MgO), at 2,800° C.[d]

According to Sosman[e]—

> Ferric oxide (Fe_2O_3) begins to dissociate, as the temperature rises, into oxygen and a solid solution containing ferrous iron; this may be considered as a solid solution of Fe_3O_4 in Fe_2O_3. At a given temperature the initial dissociation is high, but it drops rapidly as the percentage of FeO in the solid increases, passing through a range in which the pressure falls rather slowly with change of composition and finally falling rapidly to the dissociation pressure of Fe_3O_4, which is very low (less than 0.04 mm. of mercury at 1,200° C.).
>
> Fe_3O_4, in turn, dissociates into oxygen and a mixture of oxides whose character has not yet been determined.
>
> The properties of FeO are still practically unknown. The most of the recorded methods for preparing "ferrous oxide" yield only a mixture of metallic iron (or iron carbide) with an oxide whose composition falls between FeO and Fe_3O_4.

The melting point of magnetic (Fe_3O_4) is 1,580° C.[f]

RELATION OF MELTING POINTS OF ASH CONSTITUENTS TO FUSION IN THE MIXTURE.

It is evident that coal ash, being a mixture of a number of substances both crystalline and amorphous, can have no single, definite melting point; neither is the gradual softening and fusion of the ash merely the successive melting of the various ash constituents, but it is a more complicated process in which reactions involving the formation of new and more fusible compounds take place. Furthermore, the melting temperature of these newly formed compounds is influenced by solution in one another to a greater or less degree, as in the familiar example of salt and ice. Pure water freezes at 0° C. If increasing quantities of salt are added to the water the freezing point is depressed more and more until a certain minimum freezing point is reached after which further addition of salt causes the freezing point to rise again. This minimum freezing point of a mixture is known as the eutectic point, and the composition of the mixture at this point is known as the eutectic composition.

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[a] Shepard, E. S., and Rankin, G. A., The binary systems of alumina, silica, lime, and magnesia: Am. Jour. Sci., vol. 28, 1909, pp. 293–333; Ztschr. anorg. Chem. Bd. 68, 1910, pp. 370–420.

[b] Kanolt, C. W., Melting points of fire bricks: U. S. Bur. Standards Tech. Paper 10, 1912, p. 14.

[c] Kanolt, C. W., Melting points of some refractory oxides: U. S. Bur. Standards Bull., vol. 10, 1913, p. 19.

[d] Kanolt, C. W., Place quoted.

[e] Sosman, R. B., The common refractory oxides: Jour. Ind. and Eng. Chem., vol. 8, 1916, p. 985.

[f] Sosman, R. B., The common refractory oxides; Jour. Ind. and Eng. Chem., vol. 8, 1916, p. 985.

FORMATION OF COMPOUNDS AND EUTECTICS AS ILLUSTRATED IN LIME-SILICA SYSTEM.

The compounds formed on heating and cooling various mixtures of two components, with their melting and transformation points, are enumerated in figure 1, which is a concentration-temperature diagram for the lime-silica system.[a]

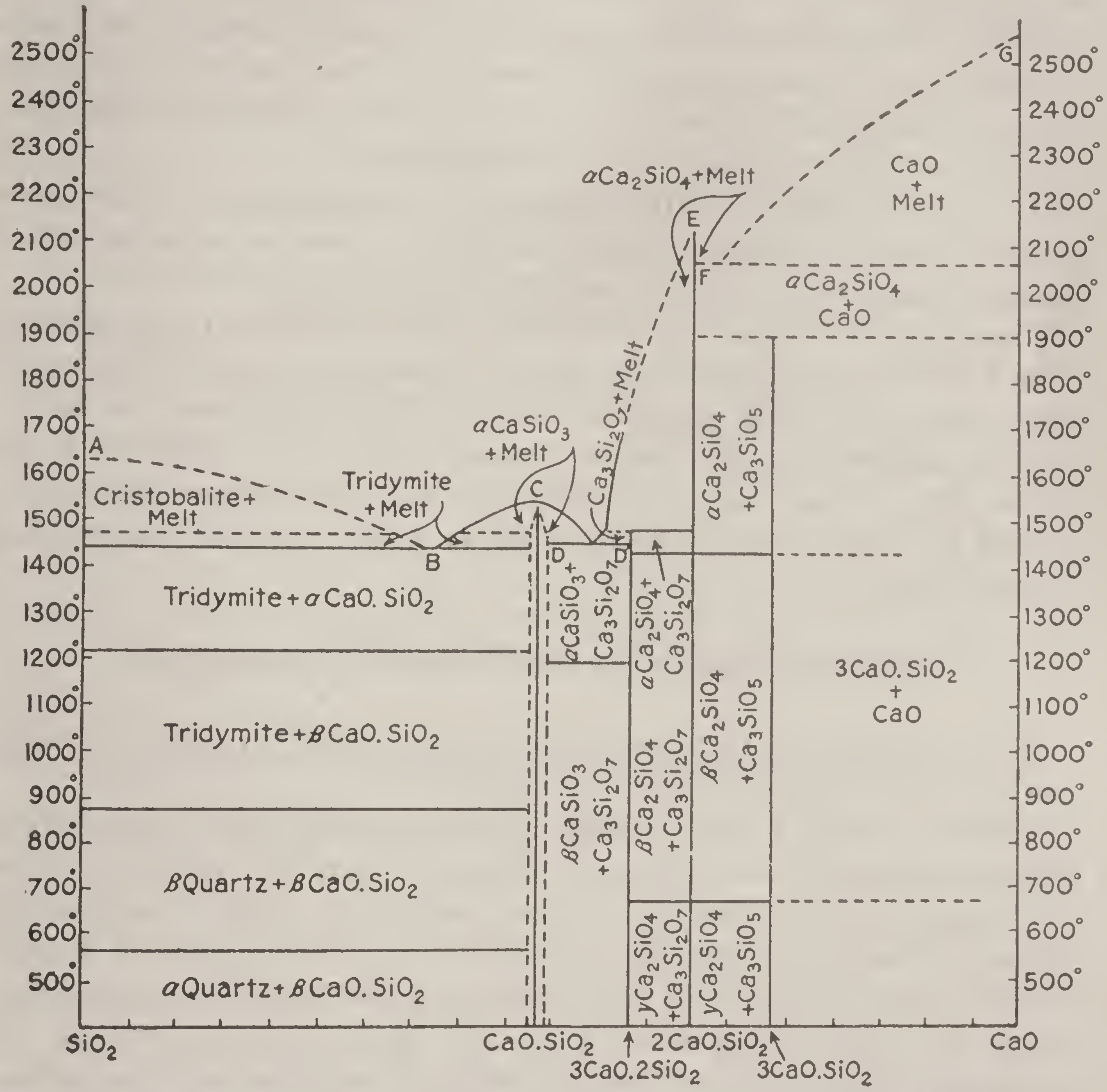

FIGURE 1.—Concentration-temperature diagram for the lime-silica system.

The melting point of pure silica in the form of cristobalite is given as 1,625° C. The melting point of pure calcium oxide (CaO) is given as 2,570° C. As increasing percentages of CaO are added to the SiO_2 the melting point is gradually depressed until at the point B a minimum or eutectic point is reached. The eutectic composition is 37 per cent CaO and 63 per cent SiO_2, and the melting point of the eutectic is 1,436° C. A further addition of lime increases the melting point until at C a maximum is reached at 1,540° C., which is the melting point of the compound pseudowollastonite ($\alpha CaSiO_3$), containing 48.2 per cent CaO and 51.8 per cent SiO_2. Any mixture of silica and lime between the points A and B and between the points B and C will show evidence of fusion when the temperature reaches 1,436° C., the melting point of the eutectic; as at this temperature that part of the mixture which is present in eutectic proportion ($37CaO$ to $63SiO_2$) will melt, leaving an excess of one or the other component unmelted. The nearer the mixture as a whole approaches the eutectic composition, the greater the proportion that melts at the eutectic temperature.

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[a] Rankin, G. A., and Wright, F. E., The ternary system $CaO-Al_2O_3-SiO_2$: Am. Jour. Sci., vol. 39, 1915, p. 5.

The point E, 2,130° C., represents the melting point of the alpha form of calcium orthosiliciate (αCa_2SiO_4). Another eutectic occurs between the points E and C, and a third between E and G.

Figure 1 shows that in mixtures of CaO and SiO_2 there are possible two compounds and three binary eutectics having definite melting points. All other ratios of CaO and SiO_2 than these five have a melting interval rather than a melting point. The interval is between the melting point of the eutectic and the temperature represented by the upper curve, A B C, at which all the material has become liquid.

DETERMINATION OF THE MELTING INTERVAL.

It is possible to determine this melting interval by the "quenching method" of the geophysical laboratory.[a]

By this method a small charge of the mixture is held at a definite temperature until equilibrium is insured, when the melted mixture is dropped suddenly into a basin of mercury, in which it is instantly cooled before any crystallization can occur. The degree of melting for the given temperature is shown by the percentage of glass found when the melted mixture is examined under the microscope. By repeating the test several times at different temperatures the melting interval may be determined.

RELATION OF MELTING INTERVAL TO CLINKER FORMATION.

The quenching method, although undoubtedly the only scientific method of determining the degree of melting at any given temperature, does not give any information regarding the viscosity of the fusion, and therefore would not in itself serve as a practical technical test of probable clinkering characteristics. As regards clinker formation, information is desired as to the temperature at which the ash may form a slag of sufficient fluidity to flow or agglomerate in the fuel bed; this temperature is usually below the temperature of complete melting as shown by the quenching method, owing to the formation of low-melting eutectics, as already described.

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[a] Day, A. L., and Sosman, R. B., The melting points of minerals in the light of recent investigations on the gas thermometer: Am. Jour. Sci., vol. 31, 1911, p. 343.

The difference between the flowing or softening temperature and the temperature of complete melting varies greatly in different ashes, depending on the relative proportion of initially melting eutectic and the viscosity of this initial slag. If this slag is very fluid it may flow readily to form clinkers, and yet contain a considerable proportion of solid constituents in suspension. On the other hand, if the ash were composed almost wholly of alumina and silica, it could be completely melted and yet not flow readily, owing to its high viscosity. Hence, any laboratory method of determining the fusibility of ash, together with the degree of clinker formation, should indicate the viscosity as well as the melting temperatures. A theoretically correct method would be one in which the viscosity-temperature curve of the melt for blast-furnace slags is determined, as has been done by one of the authors.[a] This method is not available, as when it is used the iron oxides are reduced by the graphite parts of the apparatus. Probably the simplest method of roughly determining the temperature at which the melting ash forms a slag of standard viscosity is the usual softening-temperature test, which is employed by the metallurgist in determining the "formation temperature" of slags.[b]

DETERMINATION OF SOFTENING TEMPERATURE.

Determination of the softening temperature of an ash is usually made by molding the ash in the form of a small triangular pyramid similar to a Seger cone, and noting the temperature of deformation when the cone is heated at a definite rate. The temperature at which the cone has bent over until the apex touches the base, or at which it collapses to a lump, is taken as the softening point.

Day and Shepard[c] have shown that a cone made of silicate mixtures that are capable of forming eutectics begins to weaken as soon as the eutectic begins to melt; its further progress is then governed entirely by the relative quantity of eutectic present and its viscosity after melting. If the constituents of the ash are such as to form a relatively large proportion of thinly fluid eutectic, the deformation point of the cone will lie close to the melting point of the eutectic; on the other hand, if there is a large excess of some refractory component, as silica or aluminum silicate, or if the eutectic is viscous, the excess component will form a rigid skeleton, which is not pulled down by the eutectic, the deformation point approaching more nearly the melting point of the rigid component.

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[a] Feild, A. L., A method for measuring the viscosity of blast-furnace slags at high temperatures: Tech. Paper 157, Bureau of Mines, 1916, p. 21.

[b] Fulton, C. H., Principles of metallurgy, 1st ed., 1910, p. 274.

[c] Day, A. L., and Shepard, E. S., The lime-silica series of minerals: Am. Jour. Sci., vol. 22, 1906, p. 267. Also, for a full discussion of the theory of Seger cones, see Sosman, R. B., The physical chemistry of Seger cones: Trans. Am. Cer. Soc., vol. 15, 1913, pp. 482–498.

FACTORS INFLUENCING SOFTENING TEMPERATURE.

The whole phenomenon of softening and fusion of ash is a result of chemical reaction, melting, and solution, in which the time factor and the fineness of division of the constituents are of the greatest importance. Some reactions between ash constituents, and between certain oxides, with the atmosphere in which the ash is heated take place before any melting begins. Other reactions of various velocities make their appearance after incipient fusion. Also, certain constituents have a slow rate of melting, as, for example, silica and feldspar. These substances can be heated above their melting points and yet remain in a crystalline form for some time. It is obvious, then, that the softening temperature of an ash cone must be affected by (1) the size, shape, and inclination of the cone, (2) the fineness of the ash, (3) the rate of heating, and (4) the nature of the atmosphere in which the cone is heated.

INFLUENCE OF IRON OXIDES.

Owing to the invariable presence of iron oxide in coal ash, the nature of the surrounding gases—whether oxidizing or reducing—determines, in part, the rôle which the iron shall play in the slagging reactions. In the ordinary method of preparing a coal ash by burning off the carbon in air at a red heat the iron content is practically all converted to ferric oxide (Fe_2O_3). As this oxide is heated in air it begins to dissociate at about 1,100° C.,[a] forming a solid solution containing ferrous oxide (FeO). The degree of dissociation increases with the temperature, so that ultimately all the ferric oxide is converted to magnetite (Fe_3O_4) at approximately 1,350° C.[b]. Therefore, in determining the softening temperature of an ash in air, with absolutely no reducing gases present, we are dealing with the formation of slags in which the iron component enters the reaction as ferric oxide or as a solid solution varying in composition from Fe_2O_3 to Fe_3O_4. Such slags are more refractory and more viscous than those formed by heating mixtures under conditions favoring the formation of ferrous iron. Peters[a] states that—

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[a] Sosman, R. B., and Hostetter, J. C., The oxides of iron. Solid solution in the system Fe_2O_3–Fe_3O_4: Jour. Am. Chem. Soc., vol. 38, 1916, pp. 807–833; Hostetter, J. C., and Sosman, R. B., The dissociation of ferric oxide in air: Jour. Am. Chem. Soc., vol. 38, 1916, pp. 1188–1198.

[b] Walden, P. T., On the dissociation pressure of ferric oxide: Jour. Am. Chem. Soc., vol. 30, 1908, p. 1350.

> Ferric oxide forms compounds with silica which require a high temperature for their fusion, and is consequently an unwelcome base for slags; although this substance is an almost invariable constituent of oxidized ores, it seldom makes any trouble in the blast furnace, for the reason that it is easily reduced by the fuel gases to ferrous oxide (FeO).
>
> In the more neutral atmospheres of the reverberating smelter, however, it is likely to cause delay by combining with silica, making it more difficult to melt the slag.

On the other hand, if the ash cone is heated in a reducing atmosphere of hydrogen and water vapor[b] or carbon monoxide and carbon dioxide, the ferric oxide is reduced largely to ferrous oxide, which is a much stronger fluxing agent than the higher oxides of iron. Steffe[c] found the following formation temperatures for various ferrous silicates and ferro-calcic silicates:

Formation temperatures of ferrous and ferro-calcic silicates, after Steffe.

Compound.	Formation temperature, ° C.
$4FeO,\ SiO_2$	1,158–1,174
$2FeO,\ SiO_2$	1,162–1,183
$4FeO,\ 3SiO_2$	1,162–1,181
$FeO,\ SiO_2$	1,158–1,171
$2FeO,\ 3SiO_2$	1,321–1,334
$2(FeO,\ CaO)SiO_2$	1,122–1,137

As shown by Greiner,[d] the viscosity of ferrous silicate slags is lower than that of slags in which the iron component enters the reaction in the ferric form. Hence, the conclusion seems logical that in general lower softening temperatures may be expected if, before fusion begins, the atmosphere surrounding the cone is able to reduce most of the iron to the ferrous state, but not to metallic iron, for, if the iron is reduced to the metallic state, one of the most active fluxing constituents is removed from the system. Most ashes have an excess of high-melting constituents like silica and alumina, so that any considerable reduction of iron oxide to metallic iron tends to increase the refractory effect of the silica and alumina. Hence, lower softening temperatures may be expected in tests made in gas furnaces, in which some reducing gases come in contact with the ash, whereas higher results should be found both in furnaces in which air only surrounds the cone, and in carbon or graphite electric furnaces in which strongly reducing atmospheres reduce the iron oxides to metallic iron.

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[a] Peters, E. D., Principles of copper smelting: New York, 1907, pp. 399–400.

[b] Hilpert, S., and Beyer, J., Ber. deut. chem. Gesel., Jahrg. 44, 1911, pp. 1608–1619.

[c] Steffe, Herman, Ueber die Bildungs temperaturen einiger Eisenoxydul-Kalkschlacken und einiger kalkfreien Eisenoxydul-Schlochen, deren Kenntniss für das Verschmelzen der Bleierze Bedeutung hat: Dissertation, Berlin, 1908.

[d] Greiner, E., Ueber die abhängigkeit der Viscosität in Silikat Schmelzen von ihrer Zusammensetzung: Inaugural dissertation, Jena, 1907, p. 55.

EXPERIMENTS WITH SEGER CONES.

DESCRIPTION OF FURNACES.

In studying the effect of various factors on the softening temperature of ash, the authors made tests with a selected series of coal ashes in different types of furnaces such as are in more or less common use for such tests. Each of these furnaces has its own particular atmosphere—oxidizing, reducing, or neutral—which is the principal cause

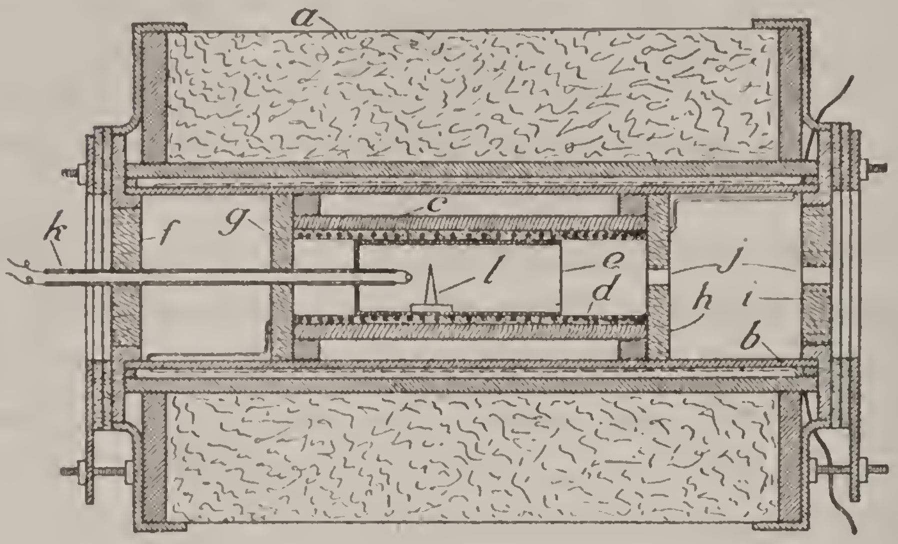

FIGURE 2.—Section through platinum-wire resistance furnace. *a*, Heraeus tube furnace, type B; *b*, porcelain tube, $2\frac{5}{8}$ inches in inside diameter and 12 inches long, wound with a platinum-foil heating element; *c*, alundum tube, $1\frac{1}{2}$ inches in inside diameter and 6 inches long, with $\frac{3}{16}$-inch wall; *d*, 17 feet of platinum wire 3/100 of an inch in diameter, wound on the inside of tube *c*, six turns to the inch in the middle and closer at the ends to compensate partly for the radiation at the ends; *e*, alundum extraction thimble No. 5811, 30 mm. in diameter and 80 mm. long; *f*, *g*, and *h*, *i*, alundum disks; *j*, observation hole; *k*, thermo element made of platinum and platinum-rhodium; *l*, ash cone.

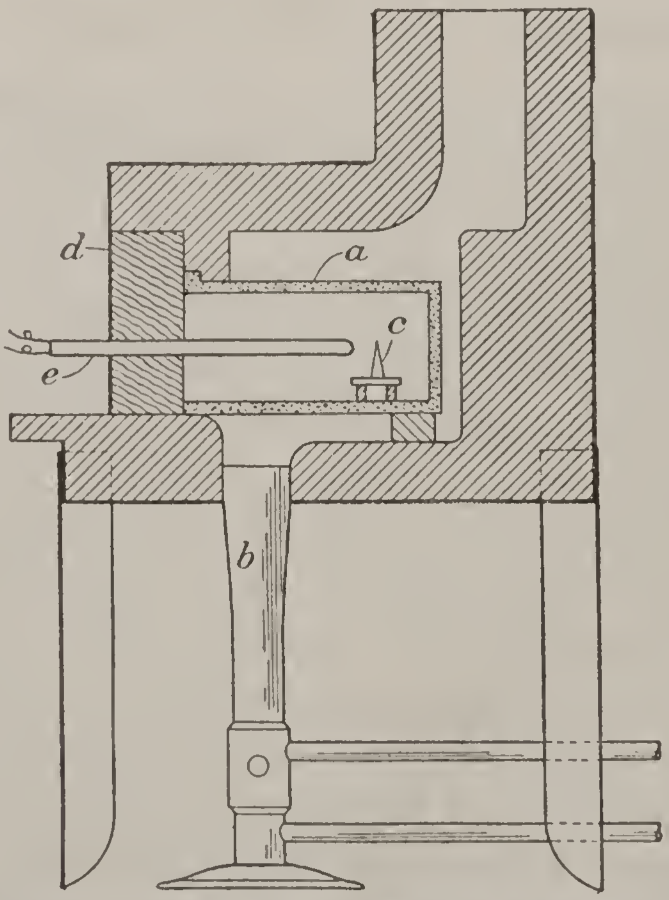

FIGURE 3.—Meker muffle furnace No. 29. *a*, Magnesite muffle; *b*, Meker blast burner; *c*, ash cone; *d*, removable fire-clay door with horizontal slot for thermocouple tube *e*.

of the large variations that were found on comparing the softening temperatures of the same sample in the different furnaces.

In the comparative tests six furnaces were used, as follows:

1. Platinum-wire resistance furnace (fig. 2), having an oxidizing atmosphere of air, with absolutely no reducing gases present.

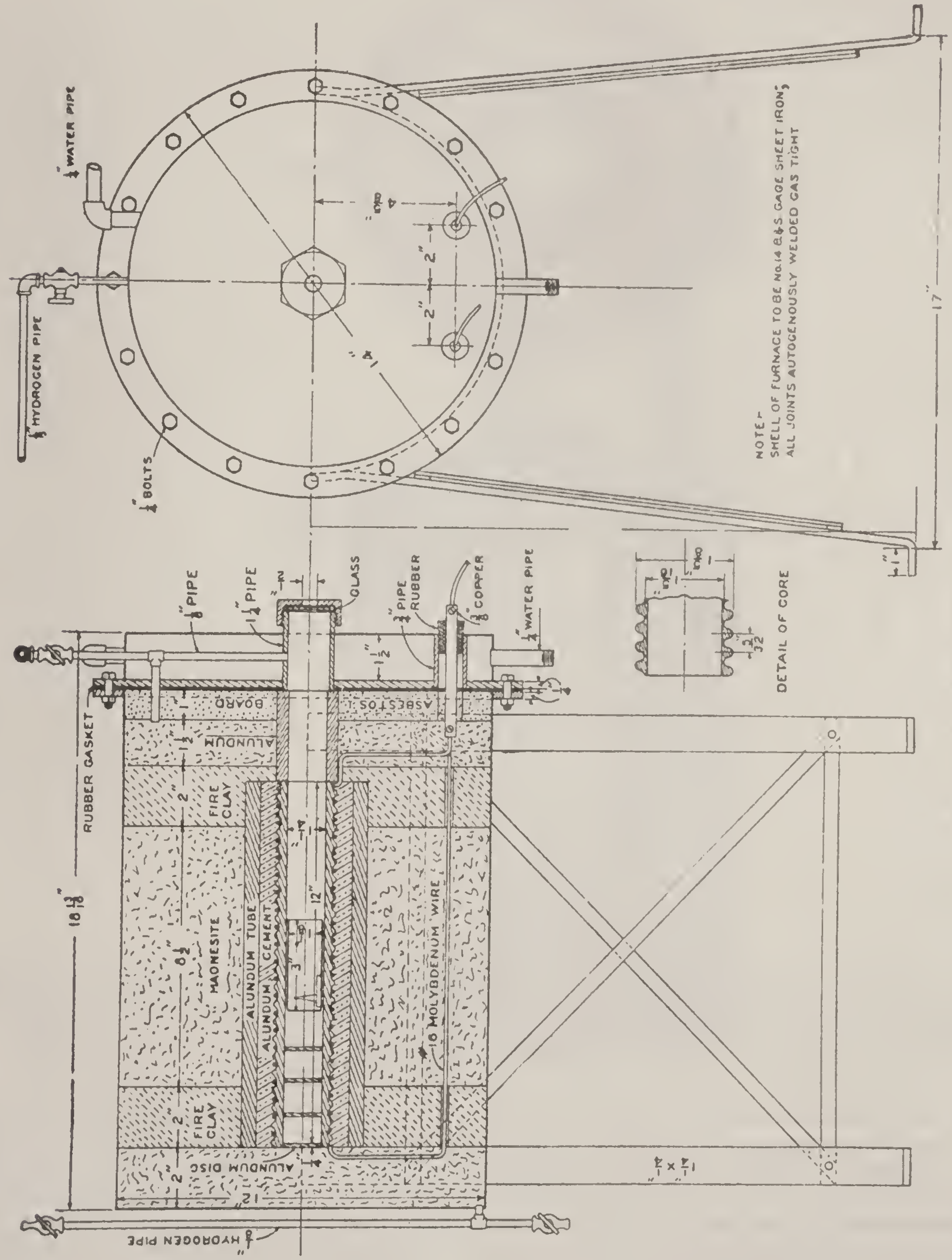

FIGURE 4.—Molybdenum furnace.

2. Meker muffle furnace No. 29 (fig. 3), having an atmosphere of air and combustion gases.

3. Muffle furnace No. 2, having an atmosphere of air and combustion gases.

4. Molybdenum-wire resistance furnace (fig. 4), having a reducing atmosphere of hydrogen.

5. Granular-carbon resistance furnace (fig. 5), having a reducing atmosphere of carbon monoxide.

6. Northrup graphite resistance furnace (fig. 6), having a reducing atmosphere of carbon monoxide.

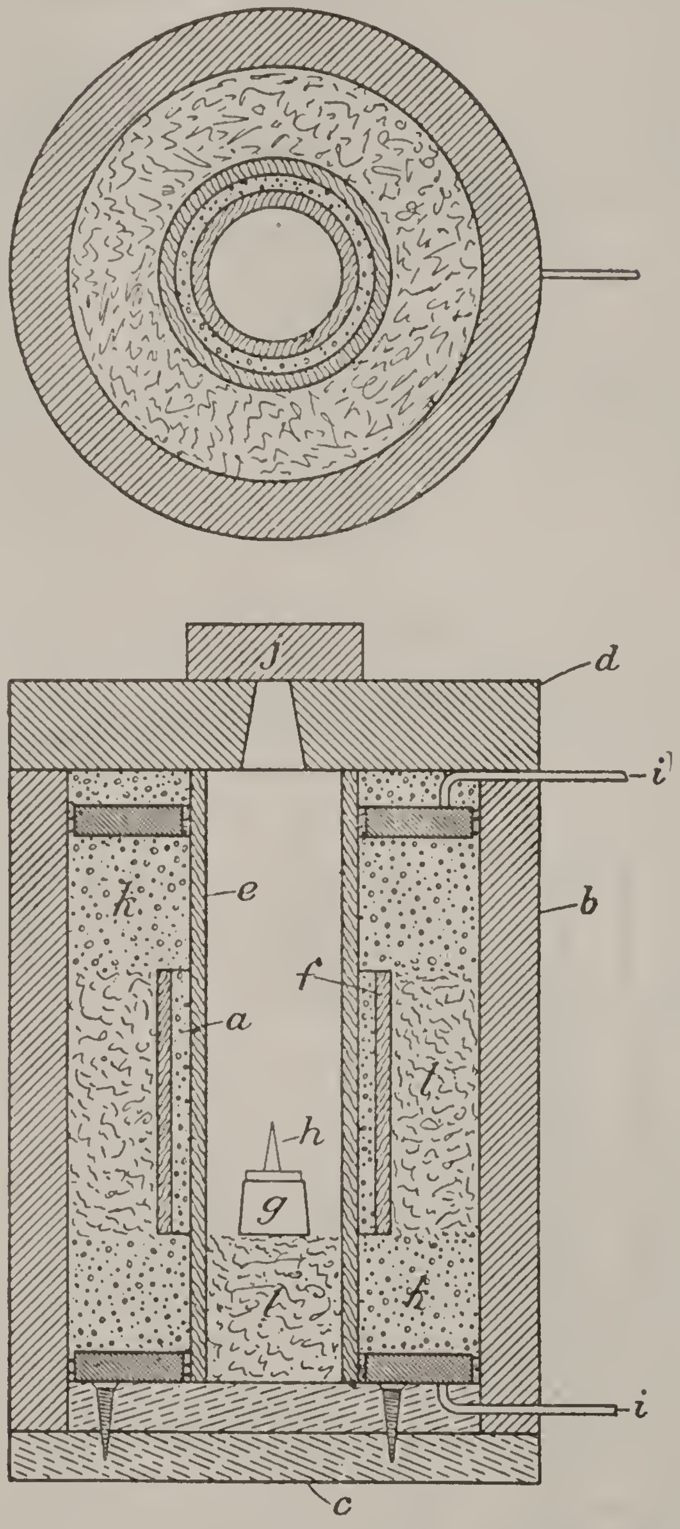

FIGURE 5.—Granular-carbon resistance furnace. *a*, Granular-carbon resister, consisting of electrode carbon crushed to pass a 6-mesh and remain on a 12-mesh screen; *b*, fire-clay cylinder, 8 inches in external diameter, 10 inches high, 1-inch wall; *c*, transit base, $\frac{3}{4}$ inch thick; *d*, fire-clay cover plate, 2 inches thick; *e*, alundum tube, 2-inch bore, 9 inches long, $\frac{1}{4}$-inch wall; *f*, alundum tube, 3-inch bore, 4 inches long, $\frac{5}{16}$-inch wall; *g*, inverted alundum crucible; *h*, ash cone; *i*, *i*, wrought-iron electrodes; *j*, fire-clay plate covering peephole; *k*, granular carbon; *l*, magnesia insulating material.

Before studying the results obtained in each furnace, as presented in the following pages, the reader is invited to notice the general method of operating the furnaces as outlined in the titles of the figures showing them.

DESCRIPTION AND ANALYSES OF COAL SAMPLES USED IN TESTS.

The description and origin of the coal samples are given in Table 1 following, and the analyses of the ash and partial analyses of the coal in Table 2 following. As may be seen from an inspection of these tables, the series of fuels tested include anthracite, bituminous, semibituminous, and subbituminous coal, lignite, and peat; the composition of the ash covers a fairly wide range, the silica content varying from 12.3 to 76 per cent; alumina from 8.6 to 34.7; ferric oxide from 3.8 to 69.7; lime from 0.6 to 18.6; and magnesia from 0.2 to 10 per cent.

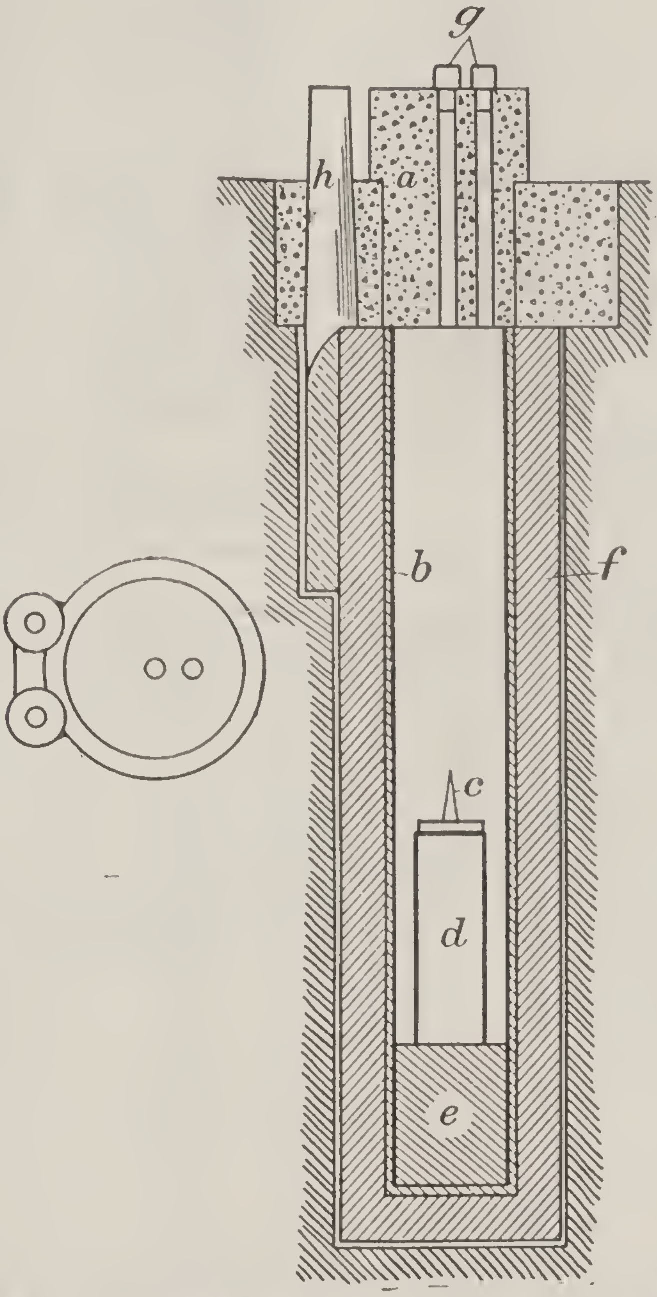

FIGURE 6.—Arrangement for softening temperature test in Northrup furnace. *a*, Refractory plug; *b*, graphite tube for protecting heater from oxidation, 1⅝ inches in internal diameter, 12¾ inches long; *c*, ash cone; *d*, inverted alundum extraction capsule, 3 inches high; *e*, solid graphite cylinder, 2 inches high; *f*, graphite heater unit; *g*, observation holes, ¼ inch in diameter; *h*, terminal of graphite heater unit.

TABLE 1.—*Character and source of coal samples.*

Sample No.	Lab. No.	Kind of coal.	Coal bed.	Location of mine.		
				Nearest town.	County.	State.
1	15840	Bituminous	Coal Creek	Fraterville	Anderson	Tenn.
2	15841	do.	American	Parrish	Walker	Ala.
3	15842	do.	Sterling	Manring	Claiborne	Tenn.
4	15843	do.	Jellico	Jellico	Campbell	Do.
5	15844	do.	Mingo	Fork Ridge	Claiborne	Do.
6	15845	do.	Coal Creek	Oliver	Morgan	Do.
7	15846	do.	Mary Lee	Red Star	Walker	Ala.
8	15847	do.	Nickel Plate	Jefferson	Jefferson	Do.
9	15848	do.	Thompson	Marvel	Bibb	Do.
10	16018	do.	Red Stone	Lemely Junction	Barbour	W. Va.
11	16019	do.	Pittsburgh	Morgantown	Monongalia	Do.
12	16243	do.	No. 5	Booneville	Warrick	Ind.
13	16583	Semibituminous	Pocahontas No. 3	Simmons	Mercer	W. Va.
14	16584	do.	do.	Bramwell	do.	Do.
15	16585	do.	do.	Elk Ridge	McDowell	Do.
16	16586	do.	do.	Big Four	do.	Do.
17	16587	do.	do.	Jenkin Jones	do.	Do.
18	16589	do.	do.	Boissevain	Tazewell	Va.
19	17081	Anthracite (buckwheat).				Pa.
20	17189	do.		Shenandoah	Schuylkill	Do.
21	17534	do.			Luzerne	Do.
22	17590	do.		Shamokin	Northumberland	Do.
23	18248	do.				Do.
24	14762	do.				Do.
25	17563	Semibituminous	Lower Kittanning	Lonaconing	Alleghany	Md.
26	[a] 7244	do.[b]	Pocahontas No. 3	Pocahontas	Tazewell	Va.
27	[a] 7305	do.[b]	do.	do.	do.	Do.
28	7308	do.[b]	do.	do.	do.	Do.
29	[c] 7309	do.[b]	do.	do.	do.	Do.
30	18193	do.	do.	do.	do.	Do.
31	18198	do.	do.	do.	do.	Do.
32	18203	do.	do.	do.	do.	Do.
33	18349	do.	do.	do.	do.	Do.
34	18350	do.	do.	do.	do.	Do.
35	18208	do.	Lower Kittanning	Ebensburg	Cambria	Pa.
36	18296	do.		Bering River Field		Alaska.
37	18297	do.		do.		Do.
38	18298	do.		do.		Do.
39	18302	do.		do.		Do.
40	18308	do.		do.		Do.
41	18310	do.		do.		Do.
42	18312	do.		do.		Do.
43	18300	do.		do.		Do.
44	18303	do.		do.		Do.
45	18306	do.		do.		Do.
46	18348	do.		do.		Do.
47	18347	do.		do.		Do.
48	18350	do.		do.		Do.
49	12690	Bituminous	Pittsburgh	Greensburg	Westmoreland	Pa.
50	[c] 12691	do.	do.	do.	do.	Do.
51	[a] 7228	do.	do.	Marianna	Washington	Do.
52	7230	do.	do.	do.	do.	Do.
53	[a] 7141	do.	do.	do.	do.	Do.
54	[c] 7458	do.	do.	Carnegie	Alleghany	Do.
55	13316	Subbituminous		Batan	Philippine Islands	
56	[c] 7548	Lignite		Rockdale	Milan	Tex.
57	[a] 7381	do.		do.	do.	Do.
58	[a] 7490	do.		Lytle	Medina	Do.
59	[a] 7494	do.		do.	do.	Do.
60	[c] 7522	do.		Calvert	Robertson	Do.
61	7534	do.		do.	do.	Do.
62	[a] 7536	do.		do.	do.	Do.
63	[a] 7159	Peat briquets		Elizabeth City	Pasquotank	N. C.
64	17559	Bituminous	Widow Kennedy		Dickinson	Va.
65	17560	do.	do.		do.	Do.
66	18121	do.	do.	Dante	Russell	Do.
67	19514	Semibituminous	Lower Kittanning or B.	Colver	Cambria	Pa.
68	19817			Stoke-on-Trent	Birchenwood Collieries.	England.
69	19818	Semibituminous	Pocahontas			
70	19841					China.
71	19842					Do.
72	20021	Bituminous			Matanuska field	Alaska.
73	20022	do.			do.	Do.
74	20023	do.			do.	Do.
75	20153	do.	Anderson	West Fork Station	Coos	Oreg.

[a] Clinker from gas producer. [b] Bone coal. [c] Clinker from boiler furnace.

TABLE 1.—*Character and source of coal samples*—Continued.

Sample No.	Lab. No.	Kind of coal.	Coal bed.	Location of mine.		
				Nearest town.	County.	State.
76	20191	Semibituminous	Pocahontas			W. Va.
77	20204	do	do			Do.
78	20217	do	do			Do.
79	20851	do	Sewell	Dunlap and Prudence.	Fayette	Do.
80	20852	do	do	do	do	Do.
81	20853	do	do	do	do	Do.
82	20913	Bituminous	Jellico		Campbell	Tenn.
83	20914	do	Coal Creek		Anderson	Do.
84	21328	Semibituminous		Huntington	Sebastian	Ark.
85	21895	Bituminous	Upper and Lower Banner.	Dante	Russell	Va.
86	[a]22665	Semibituminous	Lower Kittanning or B.	Nanty Glo	Cambria	Pa.
87	22855	Bituminous	Lower Weir-Pittsburg.	Frontenac	Crawford	Kans.
88	22988	do	Saginaw	Akron	Tuscola	Mich.
89	22991	do	No. 6	Duquoin	Perry	Ill.
90	23014	do	No. 2		Saginaw	Mich.
91	2 S.A.	Semianthracite	Spadra		Johnson	Ark.
92	3 Poca.	Semibituminous	Pocahontas No. 3		McDowell	W. Va.
93	4 S.B.	do	Sewell		Fayette	Do.
94	6 Wash	Bituminous			Pierce	Wash.
95	9 Ill.	do	Herrin or No. 6		Macaupin	Ill.
96	10 Wyo	Subbituminous	Carney		Sheridan	Wyo.
97	11 Lig.	Lignite			Williams	N. Dak.
98	13629	Bituminous	Freeport	New Kensington	Westmoreland	Pa.
99	23331	do	Pittsburgh		Washington	Do.
100	23507	do	Freeport	New Kensington	Westmoreland	Do.
101	19159	Semibituminous				W. Va.
102	10155	do				Do.
103	19161	do				Do.
104	22934	Bituminous	Lower Kittanning.	Vintondale	Cambria	Pa.
105	23136	do		Castle Gate	Carbon	Utah.
106	23137	do		Sunnyside	do	Do.
107	23138	do		Clear Creek	do	Do.
108	23581	do	Pittsburgh	Blaine Diamond	Allegheny Washington	Pa.
109	25366	do		Hale and Ladysmith.	Clearfield	Do.
110	25370	do		Norfolk No. 3	McDowell	W. Va.

[a] Clinker from gas producer.

TABLE 2.—*Analyses of ash and coal.*

Lab. No.	Sample No.	Analyses of ash.									Analyses of dry coal.		
		SiO_2.	Al_2O_3.[a]	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.	Sulphur.	Ash.	Fe_2O_3.
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P ct*	*P. ct.*	*P. ct.*	
15840	1	35.7	23.5	32.9	1.2	3.2	1.1	0.3	1.1	0.5	3.1	10.8	3.6
15841	2	47.3	34.6	9.8	1.8	1.3	.4	2.1	2.5	.1	1.7	17.4	1.7
15842	3	55.8	33.5	5.0	.9	1.5	.7	.5	2.2	.1	.8	15.6	.8
15843	4	43.3	31.4	13.6	1.2	4.2	1.4	.6	2.9	1.4	1.5	8.6	1.2
15844	5	42.2	30.6	19.0	1.2	1.3	1.0	1.3	2.9	.2	1.4	7.5	1.4
15845	6	12.3	12.2	69.7	.4	3.9	.7	.3	.6	.2	5.8	8.2	5.7
15846	7	54.1	34.7	4.5	1.5	1.2	.9	.6	2.5	.2	.7	17.7	.8
15847	8	46.8	28.7	18.0	1.4	.6	.9	1.6	2.4	.3	2.3	12.7	2.3
15848	9	54.8	27.0	7.0	1.3	4.3	1.7	.3	3.1	1.4	.6	17.5	1.2
16018	10	38.4	24.2	22.4	1.1	7.7	.9	.3	1.9	3.8	2.1	7.9	1.8
16019	11	50.4	24.0	20.4	1.4	1.7	.2	1.0	1.0	.3	2.7	10.9	2.2
16243	12	37.1	17.6	35.9	.7	3.2	.9	.4	1.8	2.3	5.8	11.5	4.1
16583	13	54.8	27.0	7.8	1.6	1.6	1.5	2.2	1.9	.5	.7	8.0	.6
16584	14	54.8	29.2	6.9	1.8	1.4	.6	1.9	2.1	1.0	.7	7.5	.5
16585	15	54.1	24.8	9.4	2.3	4.0	1.4	1.0	.8	2.8	.6	7.1	.7
16586	16	37.2	25.5	11.8	1.5	12.6	1.9	1.4	.4	5.6	.6	5.9	.7
16587	17	51.1	25.2	10.1	1.8	5.1	1.6	.8	.9	3.1	.6	6.8	.7
16589	18	51.8	25.0	9.0	2.0	4.0	1.5	1.3	.8	4.3	.7	5.6	.5
17081	19	56.1	31.4	5.0	1.9	1.0	1.0	.7	3.1	.5	.7	21.5	2.1
7244	26	68.9	21.4	4.5	2.6	2.0	.7	.6	.5		.6	17.2	.8

TABLE 2.—*Analyses of ash and coal*—Continued.

Lab. No.	Sample No.	Analyses of ash.									Analyses of dry coal.		
		SiO_2.	Al_2O_3.[a]	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.	Sulphur.	Ash.	Fe_2O_3.
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	
7305	27	69.4	21.5	4.8	2.0	1.6	.8	.5	1.4	......	.7	14.8	.7
7308	28	64.7	23.2	4.7	2.6	1.2	.8	.3	.5	1.5	.7	14.8	.7
7309	29	69.6	19.7	5.2	2.2	2.0	.4	.4	.5	......	.7	14.1	.7
12690	49	54.7	32.9	8.9	......	1.6	.9	.6	.2	......	.9	11.1	1.0
7228	51	54.5	27.0	12.1	1.8	1.5	.4	.5	1.9	......	1.7	11.4	1.4
7230	52	53.2	26.0	15.8	1.4	1.0	.7	.3	1.6	.4	1.7	11.4	1.8
7141	53	54.6	25.7	12.4	1.4	4.3	.6	.6	1.3	......	1.4	12.5	1.6
7458	54	56.8	28.2	11.3	1.2	1.0	.8	.6	1.6	......	1.8	12.3	1.4
13316	55	15.2	8.6	13.3	......	18.1	10.0	5.3	1.8	26.9	......	8.2	1.1
7548	56	47.9	23.7	5.5	2.0	18.6	3.3	.4	.2	......	1.5	16.6	.9
7381	57	46.2	24.3	6.2	1.8	18.5	3.0	.6	.5	......	1.5	15.5	1.0
7490	58	43.2	16.9	7.1	1.6	12.2	1.7	.2	.3	16.0	2.2	19.1	1.4
7494	59	52.5	21.5	8.8	1.6	14.4	2.4	.5	.2	......	2.2	19.1	1.7
7522	60	47.5	29.1	5.3	2.0	14.0	2.2	.4	.3	......	1.4	18.6	1.0
7534	61	39.3	24.8	3.8	1.9	14.9	1.7	.2	.4	12.5	1.4	16.6	.6
7536	62	46.5	28.4	5.2	2.0	16.1	2.3	.7	.2	......	1.4	16.6	.9
7159	63	76.0	11.7	4.5	1.0	2.8	3.1	1.0	.9	......	1.4	32.3	1.5
19842	71	54.1	25.8	9.0	1.0	1.9	1.2	1.6	3.8	1.4	.6	12.4	1.1
20021	72	49.9	29.1	10.0	1.1	3.7	1.4	.9	.7	2.5	.5	22.7	2.3
20204	[b] 77	59.7	26.4	7.0	......	2.6	1.3	......	(c)	......	.6	5.7	.4
20217	[b] 78	56.3	25.9	9.5	......	3.1	1.5	......	(c)	......	.6	6.2	.6
20851	79	54.3	24.6	12.4	1.1	1.4	.4	2.8	1.9	1.0	.9	7.4	.9
20913	82	48.0	28.0	11.7	1.9	3.6	1.1	1.4	1.9	2.9	1.5	8.6	1.0
20914	83	37.9	27.3	18.4	3.0	5.6	1.4	1.9	1.1	3.8	3.1	10.8	2.0
21328	[b] 84	37.4	23.3	28.8	......	3.8	3.0	......	(b)	1.1	2.3	9.5	2.7
[a] 21895	[b] 85	38.8	25.5	7.9	1.3	12.1	2.4	2.5	1.7	8.3	.6	8.6	.7
22665	[b] 86	31.0	22.7	41.9	......	2.1	.5	......	(b)	.6	3.1	8.2	3.4
22855	87	34.2	14.6	31.5	.7	7.6	.9	1.1	1.5	6.5	5.0	19.0	6.0
22991	89	52.0	25.9	13.4	1.1	2.5	.6	.8	1.6	1.8	1.8	11.2	.5
13629	98	38.2	23.4	23.5	......	5.6	1.1	5.3	1.4	2.9	2.9	12.5	2.9
23507	100	42.7	25.6	18.9	1.2	5.2	.4	.7	1.3	4.2	2.4	11.9	2.2
17559	64	50.5	29.2	9.3	2.1	1.8	2.1	.7	3.5	.5	.7	11.2	1.0
17560	65	47.8	27.0	13.6	2.1	2.4	1.7	.4	3.1	1.3	.7	8.4	1.2
18121	66	31.9	23.2	19.6	1.8	9.8	1.1	.7	2.6	8.8	1.8	7.6	1.5
19514	67	43.8	29.0	18.4	2.6	2.6	.2	.5	1.1	1.9	1.4	7.0	1.3
19817	68	48.9	24.6	10.5	2.3	4.2	2.1	1.1	2.7	3.0	.9	14.2	1.5
19818	69	49.8	25.5	9.6	1.7	3.8	2.5	1.3	3.3	2.6	.9	16.7	1.6
19841	70	......	......	......	......	......	......	......	......	......	.5	5.4	......
20022	73	51.3	25.9	8.3	2.3	5.1	3.0	1.1	1.1	2.4	.4	22.2	1.8
20023	74	49.1	28.7	7.9	1.4	5.9	1.8	.6	.8	3.3	.4	13.3	1.1
20153	75	55.9	24.9	6.5	3.5	3.5	.5	1.8	1.0	2.1	.9	21.7	1.4
20191	76	54.2	22.5	8.4	3.9	4.0	1.0	1.7	1.1	3.5	.7	5.9	.5
20852	80	46.2	27.9	18.4	2.0	2.0	.6	1.2	1.3	1.0	1.2	6.3	1.2
20853	81	47.4	27.3	15.2	2.1	1.7	1.3	1.2	2.7	.8	1.0	7.2	1.1
20913	82	......	......	......	......	......	......	......	......	......	1.0	5.5	......
20914	83	......	......	......	......	......	......	......	......	......	1.6	7.0	......
21895	85	......	......	......	......	......	......	......	......	......	.6	8.6	......
22988	88	17.8	15.6	59.8	.6	1.9	.5	1.1	.7	1.9	4.7	7.8	4.7
23014	90	40.2	26.6	14.9	1.2	7.2	.7	.8	2.4	5.8	2.4	15.2	2.3
2	91	11.0	27.5	34.6	1.4	10.7	3.1	.6	.3	9.7	1.7	9.6	3.3
3	92	59.3	21.9	5.9	6.2	1.9	.6	1.8	1.4	1.1	.6	7.4	.4
4	93	40.8	25.8	24.5	3.1	2.6	.7	.8	.7	.4	.8	3.5	.9
6	94	20.8	31.6	23.9	2.8	9.0	1.1	1.4	.8	8.6	.8	8.6	2.1
9	95	58.6	21.7	7.5	3.3	2.7	.7	2.0	2.2	1.6	4.5	11.5	.9
10	96	23.0	24.3	4.4	.7	20.6	3.0	3.8	.5	19.2	.5	5.2	.2
11	97	7.7	12.1	6.9	.3	22.9	4.2	18.2	.6	26.8	.8	8.8	.6
23331	99	46.3	22.7	10.8	1.3	8.7	.4	.7	1.1	7.5	1.5	12.3	1.3
22934	104	30.3	24.6	37.9	1.3	3.0	.2	.5	.6	2.0	2.7	7.4	2.7
23136	105	48.4	16.5	5.6	......	14.5	2.6	2.1	.7	8.8	.4	5.4	.3
23137	106	55.1	34.0	4.4	......	4.1	.1	.7	.2	1.3	.7	5.3	.2
23138	107	60.0	17.6	6.2	......	7.0	.7	1.5	.4	6.4	.6	4.1	.3
23581	108	48.0	24.7	11.6	1.8	5.8	.5	1.1	1.0	5.0	1.5	10.2	......
25366	109	42.5	31.8	19.6	1.8	2.1	.2	.8	.6	.9	1.5	8.8	......
25370	110	63.8	22.6	5.7	3.7	1.3	.7	1.2	.7	.7	.5	10.0	......

[a] P_2O_5 included with Al_2O_3. TiO_2 not determined separately but included with Al_2O_3 in all cases where no result for TiO_2 is given.

[b] Analysis is not that of coal ashed in laboratory but of ashes taken from steaming tests in which similar coal was used.

[c] Alkalies not determined.

PREPARATION OF ASH.

The coal samples were ground to 60 mesh with a crusher, rolls, and ball mill as described in Technical Paper 8.[a] The 60-mesh material was spread out on shallow, 6-inch, fire-clay roasting dishes, and reduced completely to ash with occasional stirring in a muffle furnace at a temperature not exceeding 750° C. All of the ash was finally put through a 100-mesh screen and thoroughly mixed.

PREPARATION OF CONES.

Sufficient ash to make the desired number of cones was transferred to an agate mortar, moistened with dextrin solution, and worked into a plastic mass with a spatula or pestle. After the brass mold (fig. 7) had been lubricated with vaseline the plastic material was firmly pressed into it with a knife spatula, and the surface struck off smooth to make a neat, solid triangular pyramid. The cone was then pushed out of the mold by applying a small knife blade at the base. With a little practice and proper lubrication of the mold the cone can be ejected immediately after molding without waiting for the cone to dry; a slight rounding of the bottom groove of the mold greatly facilitates removal of the cone.

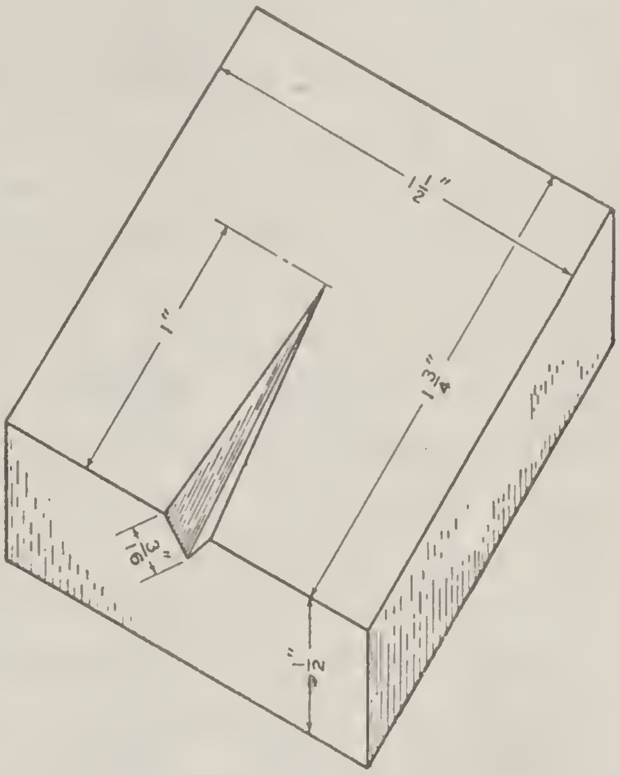

FIGURE 7.—Brass cone mold.

After the cones had dried, they were mounted in a refractory base composed of a mixture of equal parts of kaolin and alumina (Al_2O_3). This mixture was moistened with water to make it workable, and enough was taken for the base to be made and was spread on a sheet-iron plate. A small hole was made, into which the cone was set, and the base material was worked around the bottom of the cone so that it would be firmly set at the desired inclination in the base. The iron plate was then put on a hot plate, and the mounted cones were dried slowly until all water had been driven off. After the dextrin had been burned by igniting the mounted cones in a muffle, they were ready for use. In the earlier experiments the cones were made directly from the 100-mesh ash; later it was found that ash ground to an impalpable powder (or at least to pass a 200-mesh screen) made a more substantial and more easily molded cone than 100-mesh material. Four different sizes of cones were used, as follows:

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[a] Stanton, F. M., and Fieldner, A. C., Methods of analyzing coal and coke: Tech. Paper 8, Bureau of Mines, 1913, pp. 7–9.

Sizes of cones.

No. 1—Side of base, ¼ inch; height, 1 inch.
No. 2—Side of base, ¼ inch; height, 1½ inches.
No. 3—Side of base, 3/16 inch; height, 1 inch.
No. 4—Side of base, ½ inch; height, 2½ inches.

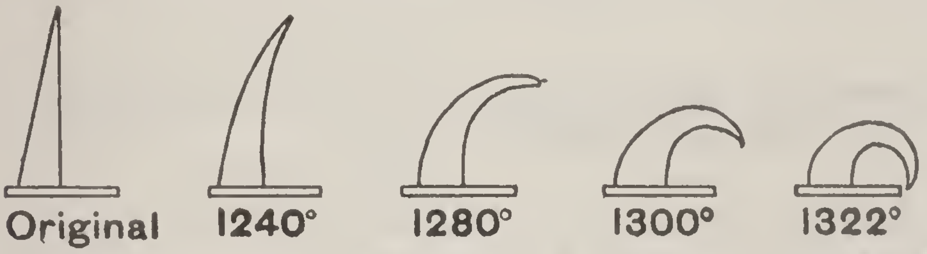
GENERAL METHOD OF HEATING.

The general procedure in making a softening-temperature determination was the same with all the furnaces used, though necessarily the details varied in the different furnaces. The test cone was put into the cold or nearly cold furnace and the furnace was heated at the rate of 10° to 15° C. per minute up to a temperature not less than 200° C. below the probable softening point, when the rate (usually 2° or 5° C. per minute) which had been adopted for that particular determination was begun. Temperature readings were taken every five minutes, and more frequently when the softening temperature was approached. Observations of the appearance of the cone were made at least as often as temperature readings were taken, special care being taken to note any deformation or warping due to shrinking before actual softening began. The point of initial softening or deformation was taken as the temperature at which the first noticeable bending, rounding at the top, or swelling of the cone took place. Warping of the cone due to shrinkage was not considered as the beginning of fusion.

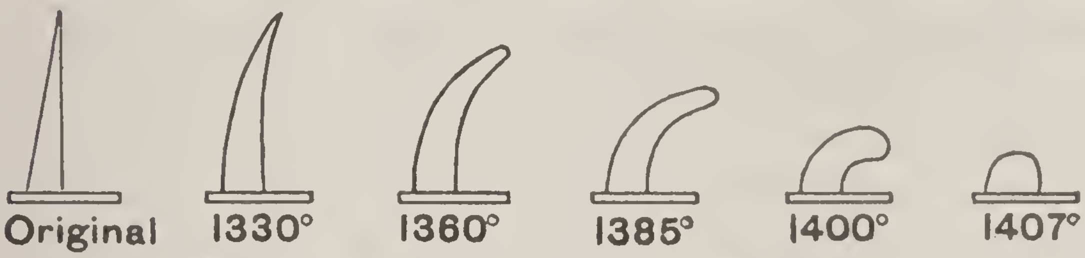
The softening point, deformation point, or "fusion point," so called, was taken as the temperature at which the apex of the cone had bent over to touch the base, or, failing to bend, had fused down to a lump or ball. Sketches (fig. 8) were made of the appearance of the cone at the initial and final deformation points and at several intermediate points of deformation, the corresponding temperatures being noted. Immediately after the temperature corresponding to complete deformation had been read, the current or other source of heat was reduced so that the appearance of the cone could be verified by examination after removal from the furnace.

TEMPERATURE MEASUREMENTS.

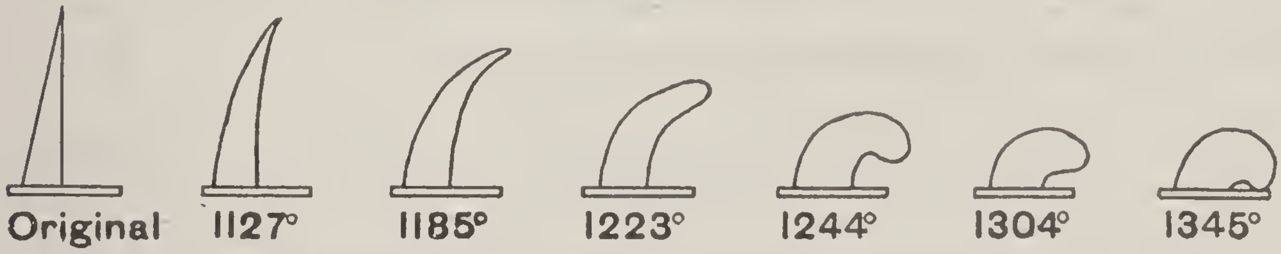
The temperature measurements were made in two ways, namely, by means of a Heraeus platinum and platinum-rhodium thermocouple with Siemens and Halske high-resistance millivoltmeter, in the platinum-resistance, Meker, and muffle No. 2 furnaces, and the down-draft ceramic kiln, and by means of a Wanner pyrometer in the molybdenum, carbon-resistance, and Northrup furnaces. The thermocouple and the millivoltmeter were standardized from time to time in the physical laboratory of the bureau under the direction of J. K. Clement, physicist, by comparison with a standard thermocouple. The cold junction was kept at the temperature of melting ice during standardization and during use in measuring temperatures. Further checks on the furnace and the pyrometer were obtained by placing crystals of pure diopside[a] in the positions occupied by the ash cones and noting the temperature of melting; usually the crystal melted between 1,381° and 1,395° C. (corrected temperature readings). The melting point of diopside is 1,391° C.[b]



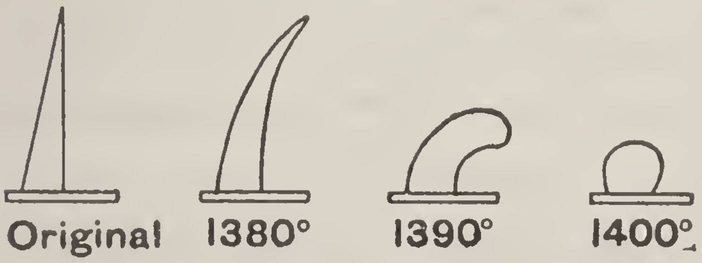
SAMPLE 16, SHOWING GOOD BEND.



SAMPLE 15, WHICH BENT AND MELTED TO A BALL.



SAMPLE 5, WHICH BENT AND PUFFED.



SAMPLE 11, WHICH BENT AND MELTED TO A BALL.

FIGURE 8.—Appearance of ash cones at various stages of softening.

The Wanner pyrometer which was originally standardized by the Reichsanstalt, was rechecked by the Bureau of Standards after most of the work described in this paper had been completed, and was found to conform to the original standardization. It was checked daily against the amyl-acetate flame. As this pyrometer was used only with those furnaces that had strongly reducing atmospheres it was possible to check the accuracy of the pyrometer readings by placing thin strips of Kahlbaum's pure copper or Kahlbaum's pure nickel in place of the cones and noting the apparent melting temperature. This was done at least once a week, and at the beginning and the end of each series of tests.

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[a] Obtained through the kindness of Dr. A. L. Day, director of the Geophysical Laboratory.

[b] Day, A. L., and Sosman, R. B., The melting points of minerals in the light of recent investigations on the gas thermometer: Am. Jour. Sci., vol. 31, 1911, p. 346.

The results obtained in these checks were usually within 10° C. of the true melting point at the copper point and within 20° C. at the nickel point.

SUMMARY OF RESULTS OBTAINED IN THE DIFFERENT FURNACES.[a]

VARIATIONS DUE TO SIZE AND SHAPE OF CONE.

Three different sizes of cones were compared in the Meker, the carbon-resistance and the molybdenum furnace, namely, cone 1, ¼ by 1 inch; cone 2, ¼-inch by 1½ inches; and cone 3, $\frac{3}{16}$-inch by 1 inch. The results, which are given in Table 3 following, indicate that cones with a wider base as compared to height, as cone 1, give higher softening temperatures than the more slender cones like cones 2 and 3. In general, the slender type of cone represented by size 3, $\frac{3}{16}$ by 1 inch, gave shorter softening intervals and less trouble from intumescence due to evolution of gases from the melting ash.

TABLE 3.—*Results of tests showing influence of size of cone on softening temperature.*

[Rate of heating, 2° C. per minute; 100-mesh ash; cones inclined 35° from vertical.]

Sample No.	Meker furnace.			Carbon furnace.			Molybdenum furnace.		
	Cone 1.	Cone 2.	Difference.	Cone 1.	Cone 2.	Difference.	Cone 1.	Cone 3.	Difference.
	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*
13	1,406	1,273	+133	1,390	1,322	+68	1,217	1,217	0
15	1,307	1,186	+121	1,311	1,250	+61	1,167	1,161	+ 6
16	1,225	1,212	+ 13	1,394	1,341	+53	1,241	1,241	0
17	1,242	1,182	+ 60	1,251	1,220	+31	1,192	1,179	+13
18	1,289	1,204	+ 85	1,316	1,250	+66	1,562	1,558	+ 4
4	1,265	1,305	− 40	1,357	1,306	+51	1,227	1,234	− 7
5	1,388	1,376	+ 12	1,401	1,356	+45	1,234	1,231	+ 3
8	1,357	1,343	+ 14	1,192	1,166	+26	1,318	1,263	+55
9	1,335	1,318	+ 17	1,327	1,249	+78	1,213	1,199	+14
1	....	....	....	1,100	1,090	+10	1,123	1,123	0
2	....	....	....	1,631	1,593	+38	1,430	1,417	+13
3	....	....	....	1,558	1,562	− 4	1,457	1,396	+61
6	....	....	....	1,207	1,179	+28	1,350	1,336	+14
7	....	....	....	1,663	1,638	+25	1,590	1,580	+10
10	....	....	....	1,187	1,167	+20	1,185	1,192	− 7
11	....	....	....	1,190	1,139	+51	1,220	1,220	0
12	....	....	....	1,088	1,088	0	1,068	1,060	+ 8
14	....	....	....	1,520	1,424	+96	1,326	1,316	+10
Average	....	....	+ 46	....	....	+41	....	....	+11

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[a] Complete experimental data and details of furnaces used are given in a previous paper, The fusibility of coal ash in various atmospheres, by A. C. Fieldner and A. E. Hall, Jour. Ind. and Eng. Chem., vol. 7, 1915, pp. 399–406, 474–481.

INFLUENCE OF FINENESS OF ASH.

Ash ground to an impalable powder tended to soften at a slightly lower temperature than 100-mesh ash, as is shown in Table 4, following. The difference averaged 3° and in no case exceeded 40° C. Ash pulverized to at least 200 mesh could be molded into more substantial cones than 100-mesh material.

TABLE 4.—*Influence of fineness of ash on softening temperature.*

[Tests made in molybdenum furnace No. 2; $\frac{3}{16}$ by 1 inch cone placed in a vertical position; rate of heating, 2° C. per minute.]

Ash sample No.	Softening point.		Difference.	Ash sample No.	Softening point.		Difference.
	100-mesh.	Impalpable powder.			100-mesh.	Impalpable powder.	
	° *C.*	° *C.*	° *C.*		° *C.*	° *C.*	° *C.*
39	1.291	1,283	+ 8	67	1,223	1,185	+38
9	1,248	1,304	−56	68	1,216	1,205	+11
42	1,340	1,313	+27	69	1,251	1,237	+14
48	1,380	1,390	−10	70	1,259	1,263	− 4
8	1,435	1,481	−46	71	1,327	1,300	+27
19	1,519	1,510	+ 9	72	1,368	1,350	+18
2	1,492	1,520	−28	73	1,380	1,370	+10
64	1,108	1,101	+ 7				
65	1,155	1,127	+28	Average			+ 3
66	1,174	1,173	+ 1				

INFLUENCE OF INCLINATION OF CONES.

Mounting the cones with a considerable inclination—25° or 45° from the vertical—led to premature deformation points in some samples, which were caused by a further bending over due to shrinkage of the cone in its base, rather than deformation due to softening and flowing of the ash. As shown in Table 5, following, differences of 64° and 65° C. were obtained with two samples. Vertical or nearly vertical cones were free from this source of error and gave the most concordant results.

TABLE 5.—*Influence of inclination of cones on softening temperature.*

[Tests made in molybdenum furnace No. 2; size of cones, $\frac{3}{16}$ by 1 inch; ash ground to an impalpable powder; rate of heating 2° C. per minute.]

Ash sample No.	Softening point.			Difference.	
	Cone inclined 45° from vertical.	Cone inclined 25° from vertical.	Cone vertical.	45° and vertical.	25° and vertical.
	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*
39	1,287	1,283	1,283	+ 4	0
42	1,295	1,313	1,304	− 9	+9
9	1,248	1,304	1,313	−65	−9
33	1,396	1,390	1,390	+ 6	0
8	1,414	1,481	1,478	−64	+3
19	1,494	1,510	1,502	− 8	+8
2	1,520	1,520	1,520	0	0
Average				−19	+2

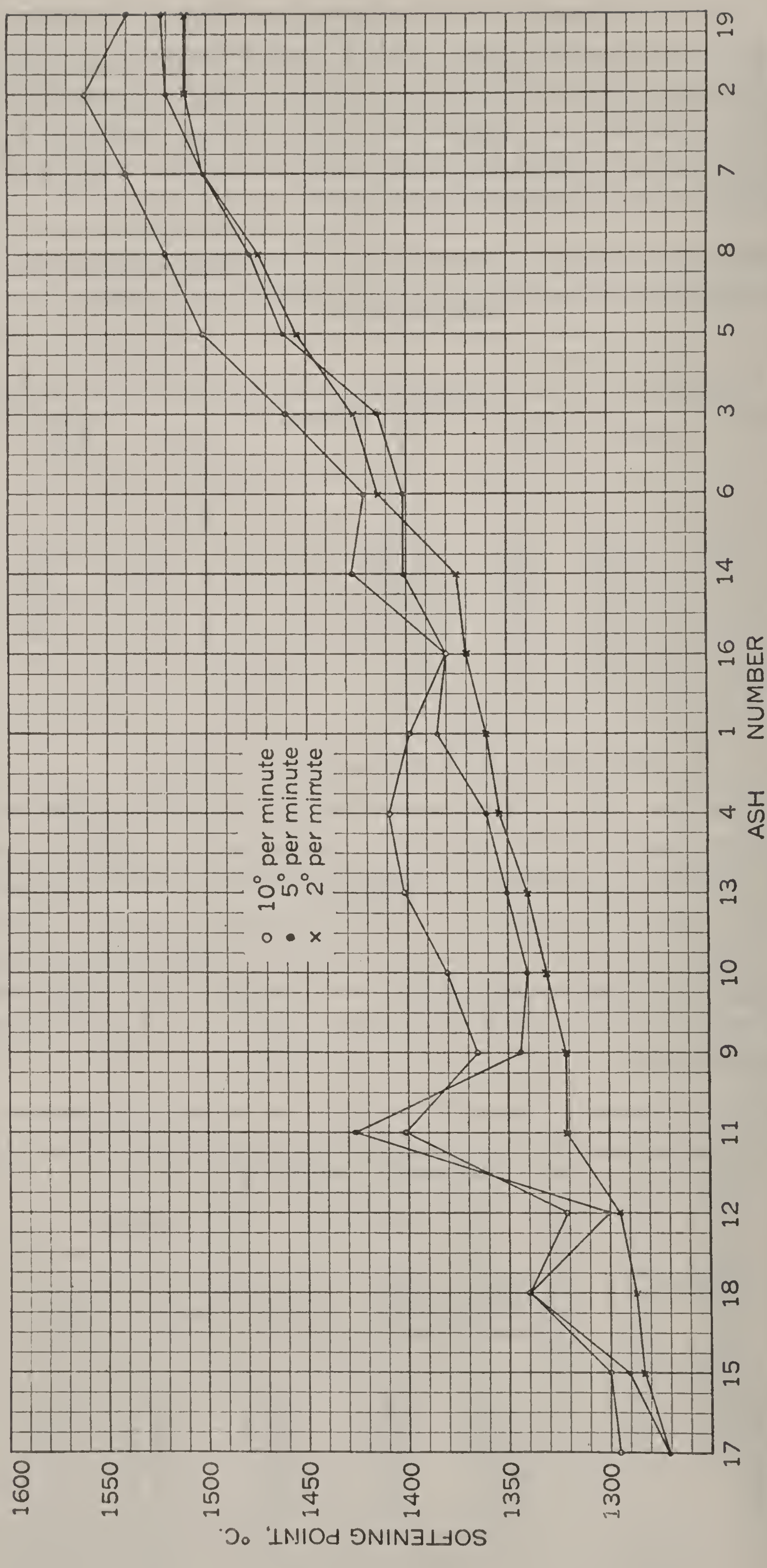

FIGURE 9.—Curves showing results of tests to determine influence of rate of heating on the softening temperature of ash cones in an atmosphere of hydrogen.

INFLUENCE OF RATE OF HEATING.

As stated previously, the rate of heating has an important bearing on the softening points of silicate mixtures made in the form of Seger cones. The fusion reactions of such mixtures require some time to approach a state of equilibrium. As it was considered probable that these reaction velocities in oxidizing atmospheres would differ from those in reducing atmospheres, tests were made at rates of heating of 10°, 5°, and 2° C. per minute in an oxidizing atmosphere of air in the platinum furnace, and in a reducing atmosphere of hydrogen in the molybdenum furnace. The softening points obtained are given in figures 9 and 10. These results show that varying the rate of heating from 2° to 10° C. per minute causes much less increase of softening temperature in oxidizing atmospheres than in reducing atmospheres of hydrogen. This would be expected because of the fact that the melting reactions in hydrogen are further complicated by reducing reactions, all of which are related to the time factor.

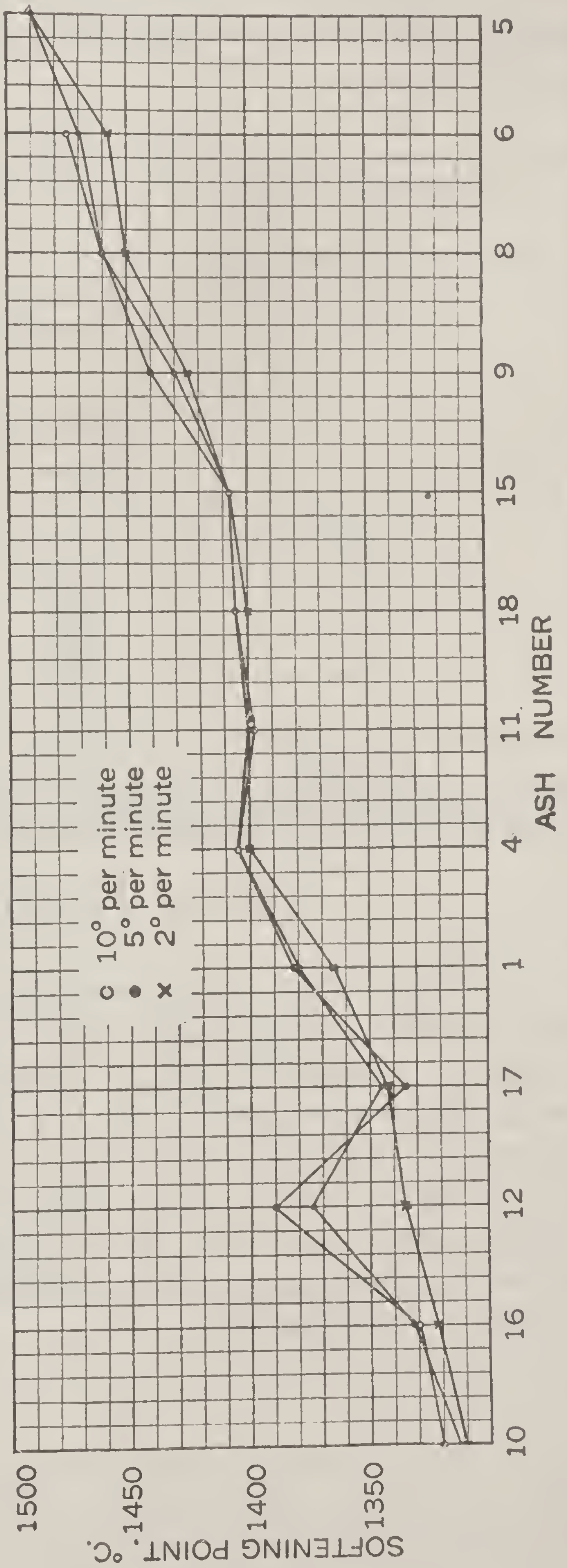

FIGURE 10.—Curves showing results of tests to determine influence of rate of heating on the softening temperature of ash cones in an atmosphere of air.

It is, of course, desirable to economize time by heating the test specimen as rapidly as is feasible. A safe maximum rate of heating in oxidizing atmospheres seems to be 10° C. per minute. However, in reducing atmospheres of hydrogen, several samples when subjected to a temperature rise of even 5° C. per minute gave results that were 50° C. higher than when subjected to a rise of 2° C. per minute. Further tests were therefore made in hydrogen, a combined 5° and 2° rate being used. The furnace temperature was increased at the rate of 5° C. per minute until deformation began; the rate was then reduced to 2° C. per minute until the cone was down. As shown by Table 6, the results approximated those obtained with the usual rate of 2° C.

TABLE 6.—*Results of tests showing comparative softening temperatures in molybdenum furnace No. 2 at two different rates of heating.*

[Ash ground to an impalpable powder; cone, 3/16 by 1 inch, placed in a vertical position.]

Ash sample No.	Softening point.		Difference.
	Temperature rise of 2° C. per minute.[a]	Temperature rise of 5° and 2° C. per minute.[b]	
	° C.	*° C.*	*° C.*
58	1,199	1,220	−21
50	1,220	1,217	+ 3
49	1,520	1,514	+ 6
55	1,380	1,370	+10
28	1,390	1,394	− 4
52	1,448	1,478	−30
61	1,365	1,370	− 5
Average			− 6

[a] Furnace temperature was increased at the rate of 2° C. per minute until cone was down.

[b] Furnace temperature was increased at the rate of 5° C. per minute until deformation began; the rate was then reduced to 2° C. per minute until the cone was down.

INFLUENCE OF DEXTRIN BINDER.

The results of a few experiments made to show the effect of not burning out the dextrin binder before placing the ash cone in the molybdenum furnace are given in Table 7 following. The maximum apparent effect was a lowering of 11° C. in the softening point. Mixing 10 per cent of dry dextrin with the ash caused a reduction of only 14° C. However, the conclusion that dextrin causes no material effect would apply only under the conditions of determining the softening point in hydrogen and water-vapor mixtures as was done in the experiments mentioned.

TABLE 7.—*Results of experiments to determine influence of dextrin binder on softening temperature of ash cones.*

[Test made in molybdenum furnace No. 2; cones $\frac{3}{16}$ in. by 1 inch, inclined 35° from vertical; 100-mesh ash; rate of increase of heating 2° C. per minute.]

Ash sample No.	Softening point.		Difference.
	Dextrin binder not ignited.	Dextrin binder ignited.	
	° *C.*	° *C.*	° *C.*
2	1,492	1,502	−10
8	1,435	1,434	+ 1
12	1,260	1,271	−11
8	1,430	1,434	− 4
Average			− 6

INFLUENCE OF DIFFERENT CONDITIONS ON SOFTENING POINT OF ASH SAMPLE 8.

Conditions.	Softening point, °C.
Cone made in usual manner, then ignited in air at 850° C.	1,434
1 per cent dry dextrin mixed with the ash; dextrin not ignited	1,427
10 per cent dry dextrin mixed with the ash; dextrin not ignited	1,420

INFLUENCE OF VARIOUS ATMOSPHERES.

None of the various factors affecting softening temperatures that have thus far been discussed seems capable of causing the exceedingly large differences in softening temperature that have been reported from different laboratories with the same samples of ash. For instance, Marks[a] has called attention to differences as great as 390° C. that were obtained with the same sample of ash by two different commercial laboratories. He himself obtained differences of 200° C. with some duplicate samples that were tested in a Meker muffle furnace and a Hoskins graphite electric furnace. As stated by Marks, these large differences were undoubtedly due to the oxidizing or reducing action of the respective furnace atmospheres. It was realized at the outset of the investigation described in this report that the reaction of the furnace gases with the iron constituent of the ash would largely determine the softening temperature. Therefore, comparative tests were made in the six different furnaces previously described in order to show the effect of various oxidizing and reducing atmospheres.

The first series of oxidizing-atmosphere tests was made in at atmosphere of air in a platinum resistance furnace (fig. 2, p. 22), in which there could be no trace of reducing gases. The second series of tests was made in a gas-heated Meker muffle furnace (fig. 3, p. 22), and the third series in a fire-clay gas muffle-furnace. As shown in figure 11, the platinum-furnace results were higher than those obtained in the gas furnaces. There was evidence of reduction by products of incomplete combustion from the fuel gases surrounding the muffles. The reduction was seemingly greater in the muffle furnace than in the gas furnaces. There was evidence of reduction by products of incomplete combustion from the fuel gases surrounding the muffles. The reduction was seemingly greater in the muffle furnace than in the Meker furnace, probably because the muffle furnace contained a rather porous fire-clay muffle, whereas the Meker furnace contained a special refractory muffle of dense texture.

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[a] Marks, L. S., The clinkering of coal: Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 205–214.

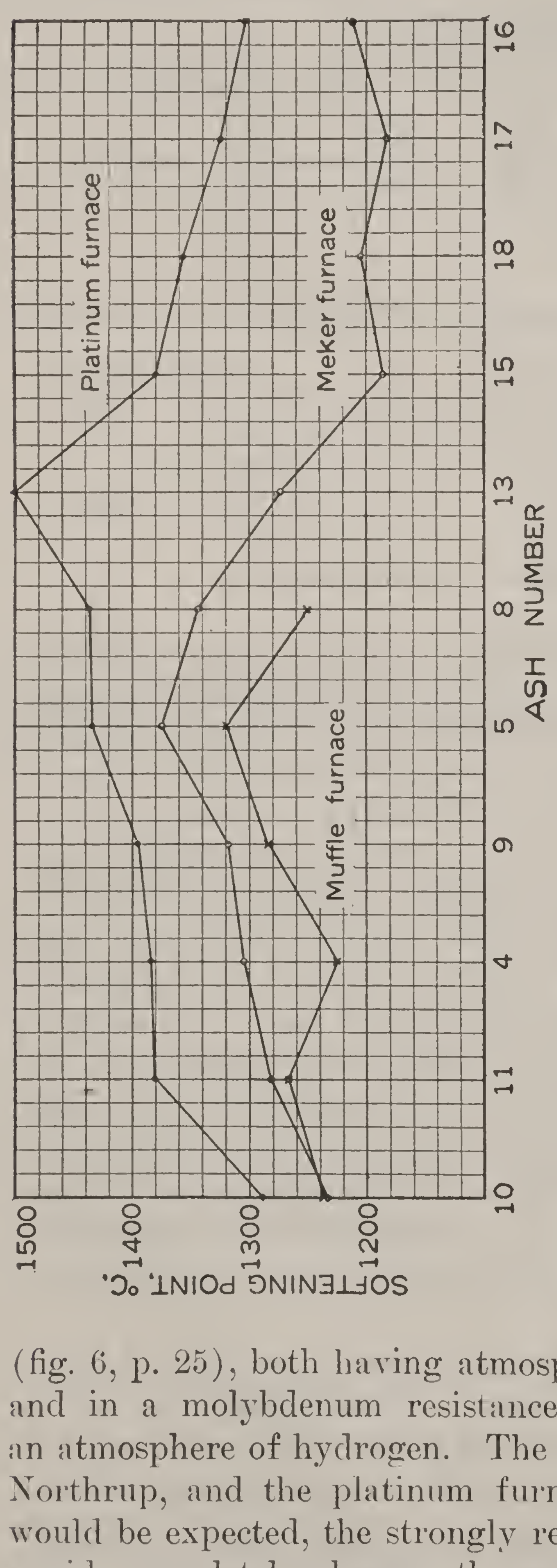


FIGURE 11.—Curves showing results of tests to determine softening temperatures in Meker furnace, muffle furnace No. 2, and platinum furnace.

From these results it is evident that softening-temperature tests made in gas furnaces must be made with the furnace operated under definite oxidizing or reducing conditions if duplicate results are to be obtained.

In the tests made in the gas furnaces there was evidence of some reduction of ferric iron to the ferrous form, but not to metallic iron. To show the effect of strongly reducing atmospheres such as would cause the formation of metallic iron, tests were made in a granular-carbon resistance furnace (fig. 5, p. 24) and a Northrup graphite furnace (fig. 6, p. 25), both having atmospheres of carbon monoxide (CO), and in a molybdenum resistance furnace (fig. 4, p. 23) having an atmosphere of hydrogen. The results obtained in the carbon, the Northrup, and the platinum furnaces are given in figure 12. As would be expected, the strongly reducing atmosphere of carbon monoxide completely changes the softening characteristics of the ash, so that results widely different from those found in the platinum furnace in air were obtained.

As both the Northrup and the granular-carbon resistance furnace produced atmospheres of carbon monoxide, similar softening temperatures were expected in the two series; that such did not prove to be the result is shown in figure 12. In 16 of 18 samples tested, the softening points in the Northrup furnace were higher than in the carbon furnace; the maximum was 396° C. and the average difference for the series was 134° C. The checks made on materials of known melting points, at various times during the course of these experiments, effectually rule out the possibility of attributing these large differences of temperature measurement. The only explanation offered by the authors is that in the carbon furnace the reduction of ferric oxide to metallic iron did not proceed as rapidly or as completely as in the Northrup furnace; in the latter the ash cone is heated in a closed tube of graphite, practically all air being excluded, so that only nitrogen, carbon monoxide (CO), and a fractional percentage of carbon dioxide (CO_2) surrounds the ash at temperatures above 1,100° C. Such an atmosphere would reduce the ferric oxide to metallic iron and thus prevent the formation of a low-melting ferrous silicate eutectic. On the other hand, the carbon furnace had an alundum tube between the carbon resistor and the cone, and it also permitted air to leak into the heating space as shown by the appearance of carbon monoxide flames at the peephole and around the cover plate. Sufficient carbon dioxide may therefore have been present to retard the reduction to metallic iron, thus leaving enough of the strongly fluxing ferrous oxide in the ash to form a low-melting fluid eutectic which caused the cone to collapse at a comparatively low temperature. Examination of polished sections of fused cones from both furnaces disclosed more metallic iron in those from the Northrup furnace.

Figure 13 shows the results obtained in the series of tests in the hydrogen atmosphere of the molybdenum furnace as compared with the results of the tests in the platinum and the Northrup furnaces. It is quite evident that the softening temperature in hydrogen is different from that in air or carbon monoxide.

The softening temperatures obtained in the six different furnaces are given in Table 8 following. The maximum differences range from 134° to 396° C. As regards 14 samples that softened below 1,500° C. in the platinum furnace in air, 9 gave their highest results in the strongly reducing carbon monoxide atmosphere of the Northrup furnace (see fig. 12); and 5 gave their highest results in the oxidizing atmosphere of the platinum furnace. The lowest softening points in 18 samples were distributed among the different furnaces as fol-

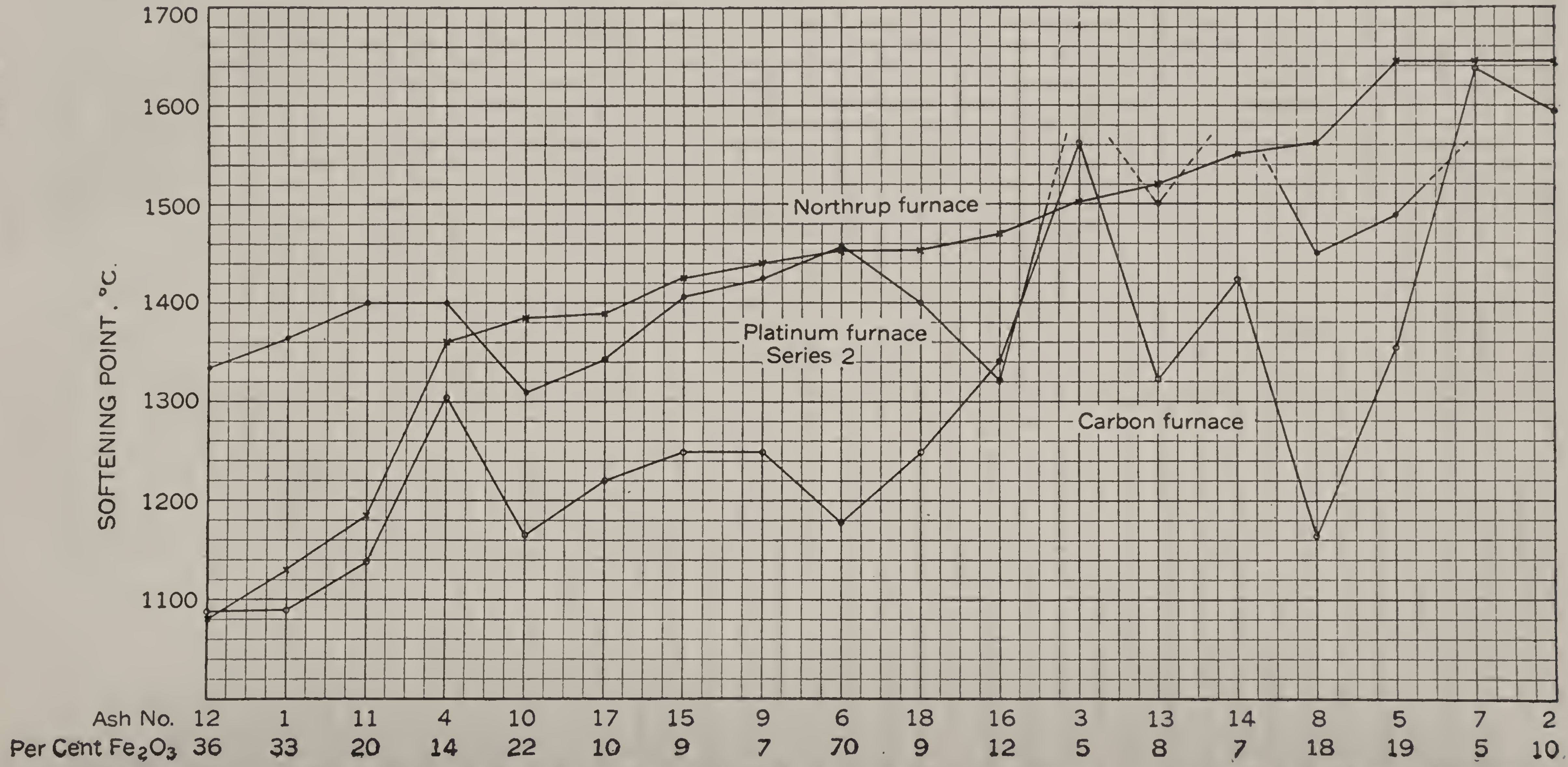


FIGURE 12.—Curves showing results of tests to determine softening temperatures in Northrup, carbon-resistance, and platinum furnaces.

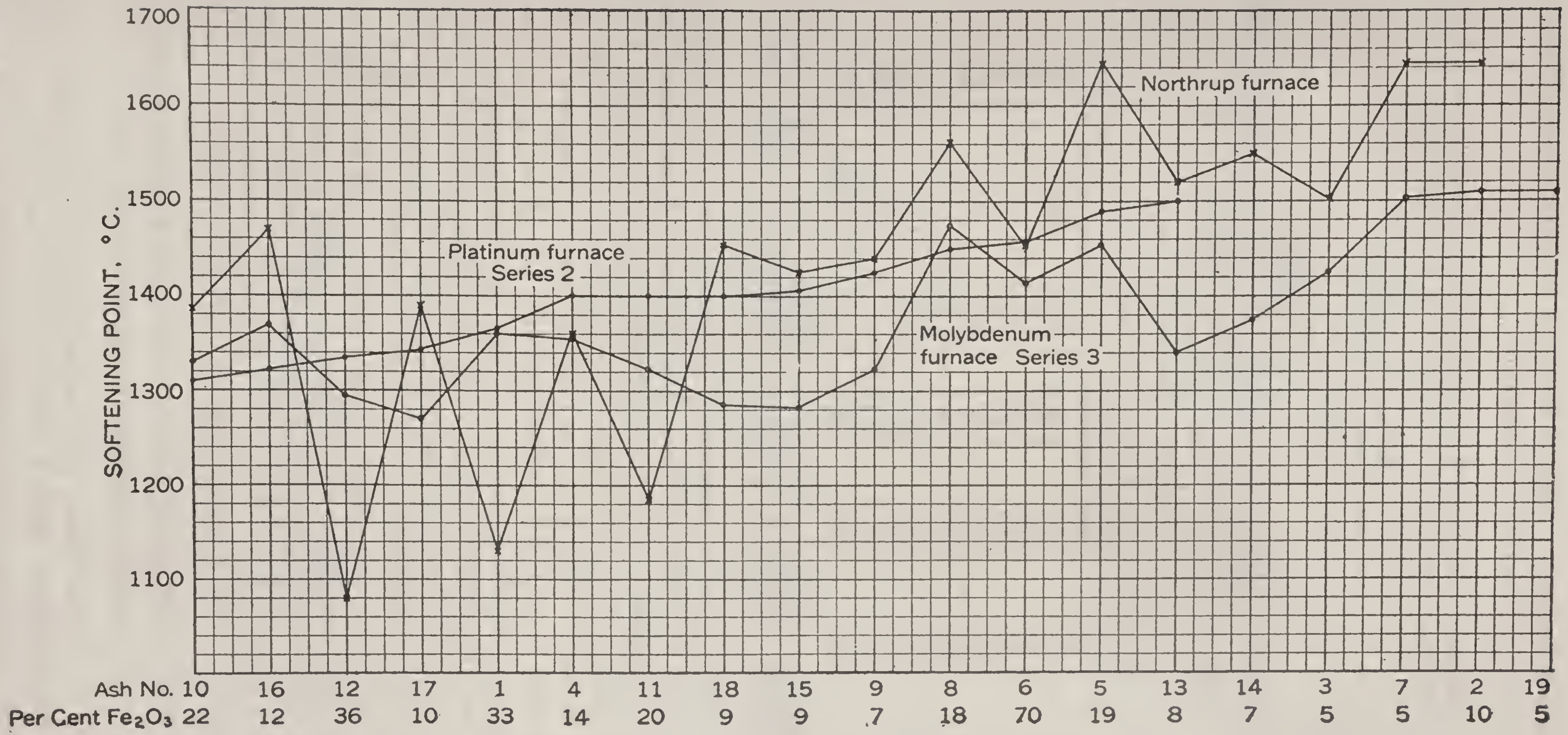


FIGURE 13.—Curves showing results of tests to determine softening temperatures in Northrup, platinum, and molybdenum furnaces.

lows: Carbon furnace, 6; Meker furnace, 5; molybdenum furnace, 4; muffle furnace, 2; Northrup furnace, 1.

TABLE 8.—*Results of tests to determine influence of various atmospheres on softening temperatures of ash cones in different furnaces.*

[Rate of heating, 2° C. per minute.]

Ash sample No.	Softening point.						Maximum difference.
	Northrup furnace,[a] in atmosphere of CO+N.	Carbon furnace,[b] in atmosphere of $CO+CO_2+N$.	Molybdenum furnace,[b] in atmosphere of H.	Muffle furnace,[b] in atmosphere of air+combustion gases.	Meker furnace,[b] in atmosphere of air+combustion gases.	Platinum furnace,[a] in atmosphere of air.	
	°C.	°C.	°C.	°C.	°C.	°C.	°C.
1	1,131	1,090	1,360	1,192	....	1,365	275
2	1,645	1,593	1,511	....	....	Above 1,500	134
3	1,502	1,562	1,427	....	....	Above 1,500	135
4	1,360	1,306	1,355	1,226	1,305	1,400	174
5	1,645	1,356	1,455	1,320	1,376	1,490	325
6	1,455	1,179	1,414	1,318	....	1,458	279
7	1,645	1,638	1,502	....	....	Above 1,500	143
8	1,562	1,166	1,474	1,251	1,343	1,450	396
9	1,440	1,249	1,322	1,284	1,318	1,425	191
10	1,385	1,167	1,331	1,199	....	1,310	218
11	1,185	1,139	1,322	1,214	....	1,400	261
12	1,080	1,088	1,295	1,182	....	1,335	255
13	1,520	1,322	1,340	....	1,273	1,500	247
14	1,551	1,424	1,375	....	1,417	Above 1,500	176
15	1,427	1,250	1,283	....	1,186	1,407	241
16	1,470	1,341	1,370	....	1,212	1,322	258
17	1,390	1,220	1,271	....	1,182	1,343	208
18	1,455	1,250	1,287	....	1,204	1,400	251

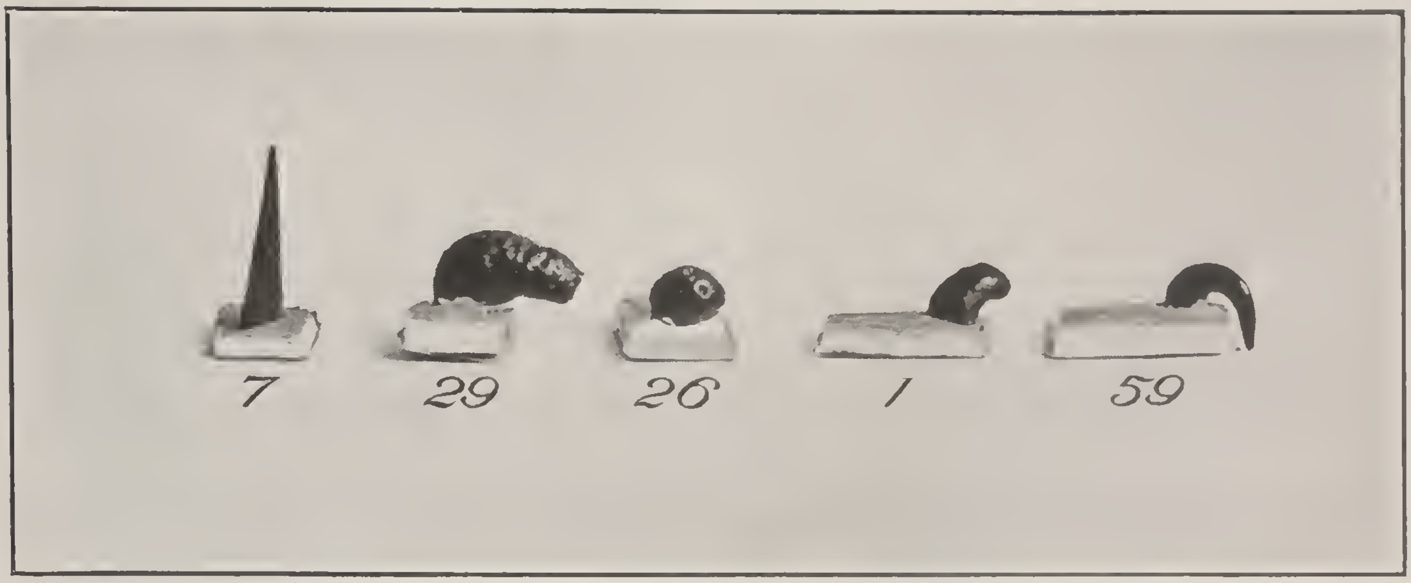
[a] Ash ground to an impalpable powder; cones $\frac{3}{16}$ by 1 inch, placed in vertical position.
[b] Ash 100 mesh; cones $\frac{1}{4}$ inch by $1\frac{1}{2}$ inches, inclined 35° from vertical.

There seems to be no consistent relation between the results obtained in the different furnace atmospheres. Some ashes have a higher softening point in an oxidizing atmosphere; others in a reducing atmosphere of carbon monoxide. The only general conclusion that can be made regarding the effect of the atmosphere is that a given ash will have its highest temperature of softening in either an atmosphere of air, free from all traces of reducing gases, or in a strongly reducing atmosphere in which the concentration of reducing gases is so great that the iron constituents of the ash are largely reduced to the metallic form. Between these two extremes of oxidation and reduction, there seems to be a condition of slight reduction by mixed gases such as was found in the muffle furnaces, in which reduction of ferric oxide proceeded until the oxide was largely in the ferrous state. On account of its strongly fluxing action ferrous oxide greatly lowers the temperature of slag formation.

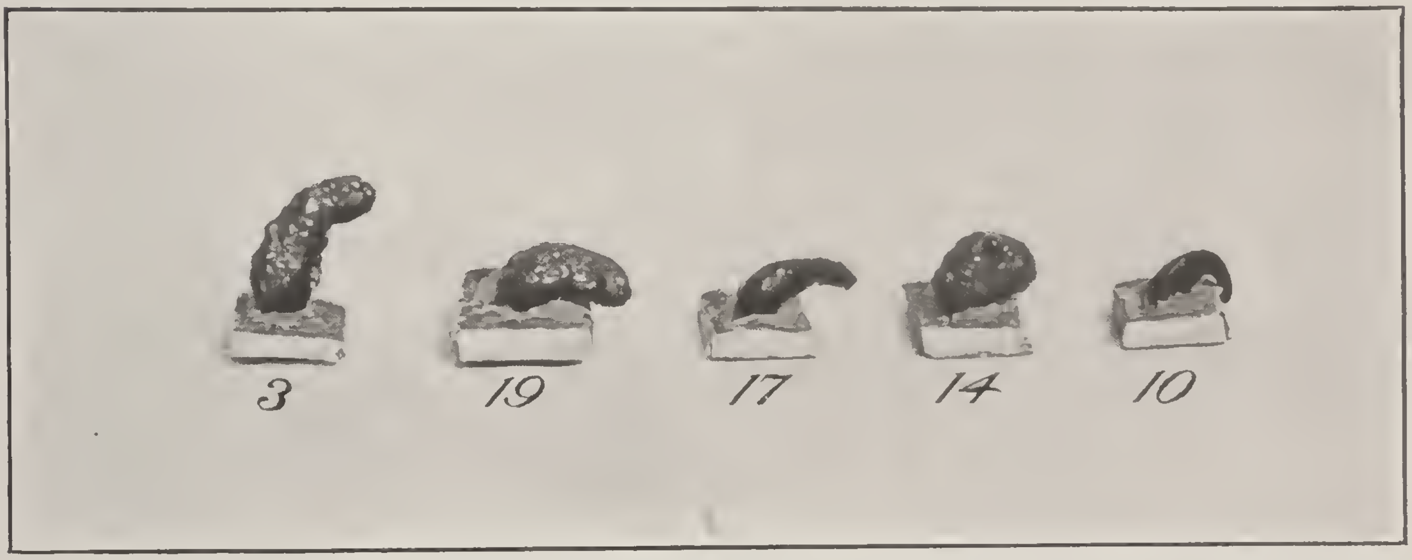
APPEARANCE OF CONES SOFTENED IN VARIOUS ATMOSPHERES.

PLATINUM FURNACE, ATMOSPHERE OF AIR.

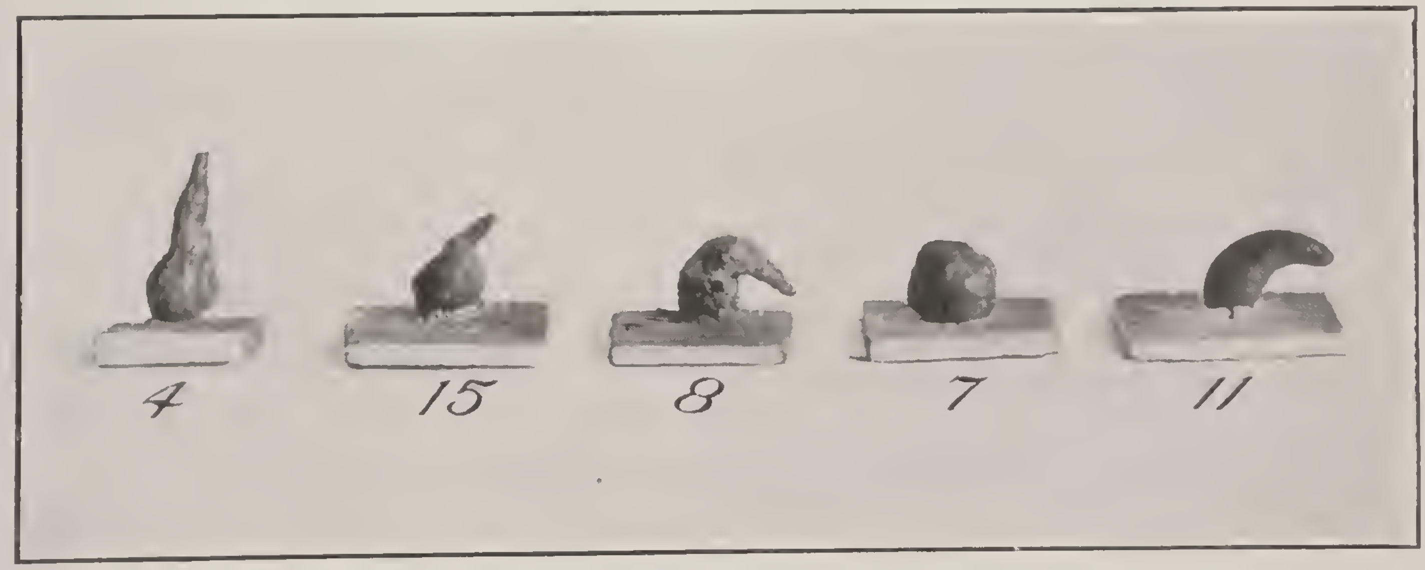
In Plate I, *A*, are shown the various shapes into which cones of different ashes deform when heated in air in the platinum furnace. The manner of deformation is somewhat characteristic for each indi-



A. TYPICAL FORMS OF CONES FUSED IN AIR IN THE PLATINUM FURNACE.



B. TYPICAL FORMS OF CONES FUSED IN HYDROGEN IN THE MOLYBDENUM FURNACE.



C. TYPICAL FORMS OF CONES FUSED IN CARBON MONOXIDE IN THE NORTHRUP FURNACE.

vidual ash. No. 59 illustrates the typical form of bending of Seger cones; No. 26 illustrates the type that fuses to a lump; No. 29 is representative of the intumescing type; No. 7 is an ash which did not soften at 1,500°. The color of the fused cones varied from a buff, or reddish brown, to a glossy black. The black cones were magnetic.

MOLYBDENUM FURNACE, ATMOSPHERE OF HYDROGEN.

Plate I, *B*, illustrates typical forms from the molybdenum furnace (hydrogen atmosphere). Owing to the reducing atmosphere, these cones almost always developed a pronounced vesicular structure. The swelling due to the evolution of gas in some cones, as in ash No. 3, prevented the cones from bending in the normal manner. This is one of the reasons for the somewhat larger variations between duplicate determinations in reducing atmospheres. The fused cones were usually light to dark gray or grayish black; they had a vitreous luster and invariably showed iron globules when examined under the microscope.

NORTHRUP FURNACE, ATMOSPHERE OF CARBON MONOXIDE AND NITROGEN.

The cones fused in the Northrup furnace (Pl. I, *C*) (CO atmosphere) varied from a dark-gray to a brownish-black or metallic-black color. The surface lacked the vitreous luster of the cones fused in air or hydrogen. Carbon was deposited on the cones and the bases under the cones in increasing quantity as the temperature exceeded 1,400° C., as shown by the blackened base of ash No. 8 (Pl. I, *C*), which was heated to 1,550° C.

Another peculiarity in the cones tested in the Northrup furnace was the softening and swelling of the thick bottom parts of some of the cones before the apexes began to soften. Ash No. 4 began swelling at the base at 1,200° C.; heating was continued at a temperature increase of 2° C. per minute until a temperature of 1,375° C. was reached, when the current was interrupted and the furnace allowed to cool. As shown in Plate I, *C*, fusible constituents appeared to separate and run down the sides of each cone, leaving a refractory skeleton standing. However, when the cone was broken, the irregular contour appeared to be due to the evolution and expansion of gases in the interior of the softening slag. The expanded bottom was a thin-shelled bubble of slag.

The first perceptible swelling of the lower part of ash No. 8 took place at 392° C. below the temperature at which it finally bent over in the position shown in Plate I, *C*.

The earlier fusion of the thick part of the cone may be due to the inability of the carbon monoxide to penetrate and reduce to metallic iron the iron oxide in the interior as rapidly as in the thin upper part. This phenomenon of irregular softening was noted only in the Northrup and the carbon-resistance furnaces.

STANDARD METHODS FOR DETERMINING SOFTENING TEMPERATURE OF ASH.

A standard method for determining the softening temperature of ash may now be considered. Obviously, the statement that an ash has a "fusing" or softening temperature of 1,500° C. means nothing unless the exact conditions of making the test are defined. The method must therefore be carefully standardized in all its details before directly comparable results may be obtained by different workers.

The most important consideration is the atmosphere in which the test shall be made. In order to obtain results that can be duplicated in different laboratories, the softening temperature in air, as was determined in the platinum furnace, is undoubtedly the most satisfactory method, but, unfortunately, ash forming in a fuel bed is not free from the effect of reducing gases. Consequently, clinkering may take place under conditions of partial reduction similar to those that prevailed when the low softening points were obtained in the experiments described. Therefore the danger point as regards clinker formation is the lowest possible softening temperature of an ash under such conditions of reduction as are probable in a fuel bed.

The problem resolves itself into an investigation of the effects of various mixtures of a reducing gas and its product of oxidation on the softening temperature with a view to finding that mixture which will give the lowest softening temperature. The two principal reducing gases to be considered are hydrogen and carbon monoxide, the respective oxidation products being water and carbon dioxide.

FUSIBILITY OF COAL ASH IN MIXTURES OF HYDROGEN AND WATER VAPOR.

THEORETICAL CONSIDERATIONS.

The equilibrium between hydrogen, water vapor, Fe, and Fe_3O_4 according to the reaction,

$$3Fe+4H_2O \rightleftarrows Fe_3O_4+4H_2 \qquad (1)$$

was first studied by Deville[a] at various temperatures between 200° and 1,000° C., and later with greater precision by Preuner[b] at three temperatures. Preuner found the following mean values for the equilibrium constant:

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[a] Deville, H. Sainte-Claire, Action de l'eau sur le fer et le hydrogène sur l'oxyde de fer: Compt. rend., t. 70, 1870, pp. 1105–1111 and 1201–1205.

[b] Preuner, G., Das Gleichgewicht zwischen Eisen, Eisenoxyduloxyd, Wasserstoff, und Wasserdampf: Ztschr. phys. Chem., Bd. 47, 1904, pp. 385–417.

Equilibrium data of Preuner.

Temperature of solid phase, °C.	pH_2O.
900	0.69
1025–1050	.78
1150	.86

Chaudron[a] has since reported two series of equilibria of iron and its oxides in hydrogen and water vapor corresponding to the following reactions:

$$Fe+H_2O \rightleftarrows FeO+H_2 \qquad (2)$$
$$3FeO+H_2O \rightleftarrows Fe_3O_4+H_2 \qquad (3)$$

His values for the equilibrium constants are as follows:

Equilibrium data of Chaudron.

Reaction 2.		Reaction 3.	
Temperature of solid phase.	$\frac{pH_2O}{pH_2}$	Temperature of solid phase.	$\frac{pH_2O}{pH_2}$
°*C.*		°*C*	
360	0.12	440	0.24
440	.18	480	.32
500	.23	500	.33
550	.32	615	.54
600	.31	700	1.00
630	.36	710	1.00
660	.39	800	1.35
760	.55	860	2.00
875	.74		

No similar data are available on the behavior of ferric oxide (Fe_2O_3); hence these data for magnetite include the only available information as to the probable or possible phases in which the iron oxide of coal ash may exist in various mixtures of hydrogen and water vapor at the sintering temperature of ash.[b]

In figure 14 are plotted the equilibrium data of Preuner and of Chaudron. Assuming that Fe_2O_3 reacts as regards reduction in a similar manner as does Fe_3O_4, it is evident that at temperatures up to 1,150° C. no reduction to metallic iron can take place in a mixture of equal parts by volume of hydrogen and water vapor. At a temperature of 850° C., according to Chaudron's figures, FeO would be the stable solid phase in mixtures containing not less than 43 or more than 63 per cent of water vapor, it being assumed that a gas mixture of constant composition is passed over the heated oxide continuously, sweeping the gaseous products of the reaction out of the apparatus.

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[a] Chaudron, G., Réactions réversibles de l'eau sur le fer et sur l'oxyde ferreux: Compt. rend., t. 159, 1914, pp. 237–239.

[b] Recent investigations by Sosman and his coworkers at the geophysical laboratory have shown that the equilibrium relationships of the iron oxides are further complicated by solid solution. In the system Fe_2O_3–Fe_3O_4 there is a continuous solid solution series from Fe_2O_3 to Fe_3O_4.

It seems possible, therefore, that a minimum softening temperature of coal ash may be found in a mixture approximating equal parts by volume of hydrogen and water vapor, provided the heating is sufficiently slow to allow the reaction to approach equilibrium at the various temperatures.

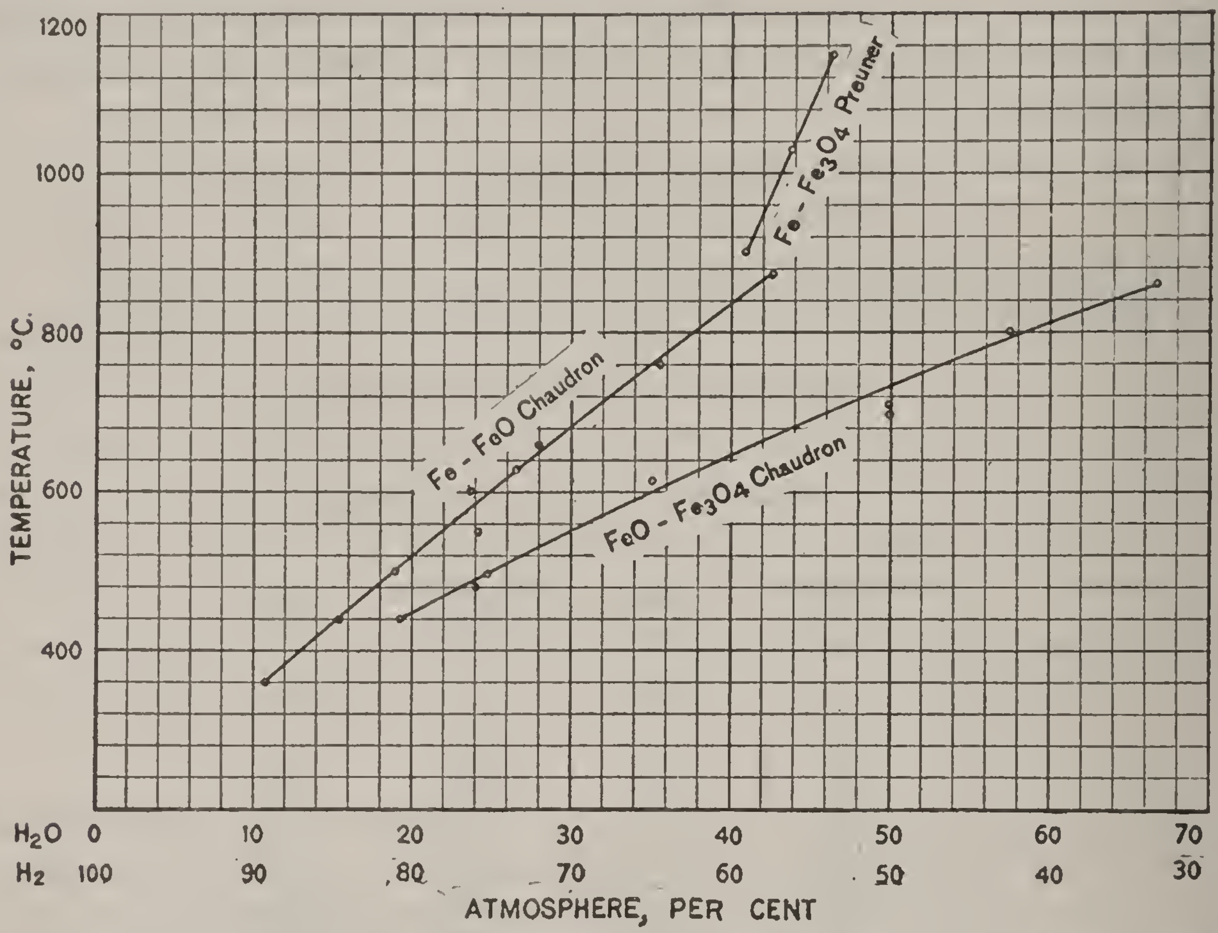


FIGURE 14.—Equilibrium diagram of the system H_2, H_2O, Fe, FeO, and Fe_3O_4.

OUTLINE OF EXPERIMENTS.

DESCRIPTION OF FURNACE.

The arrangement of the furnace and of the saturator used in the authors' tests is shown in Plate II, and figures 15 and 16. A Heræus platinum-foil resistance furnace which happened to be available was equipped with an extra heater of $\frac{3}{100}$-inch platinum wire, *a* (fig. 15),

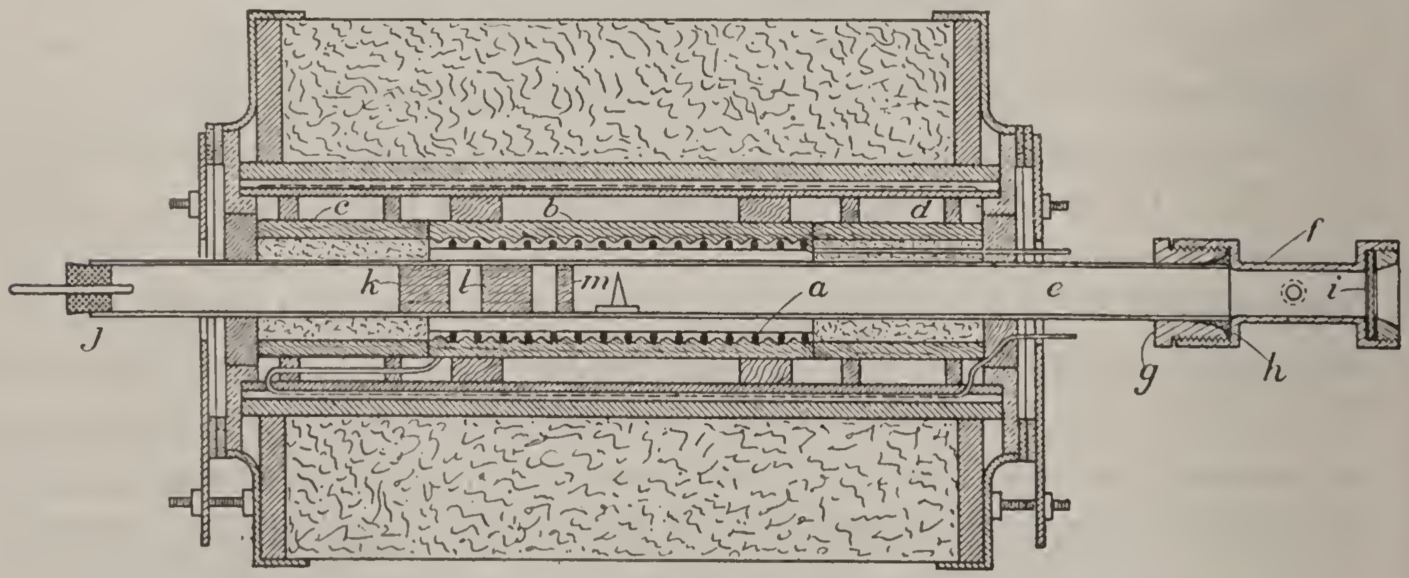
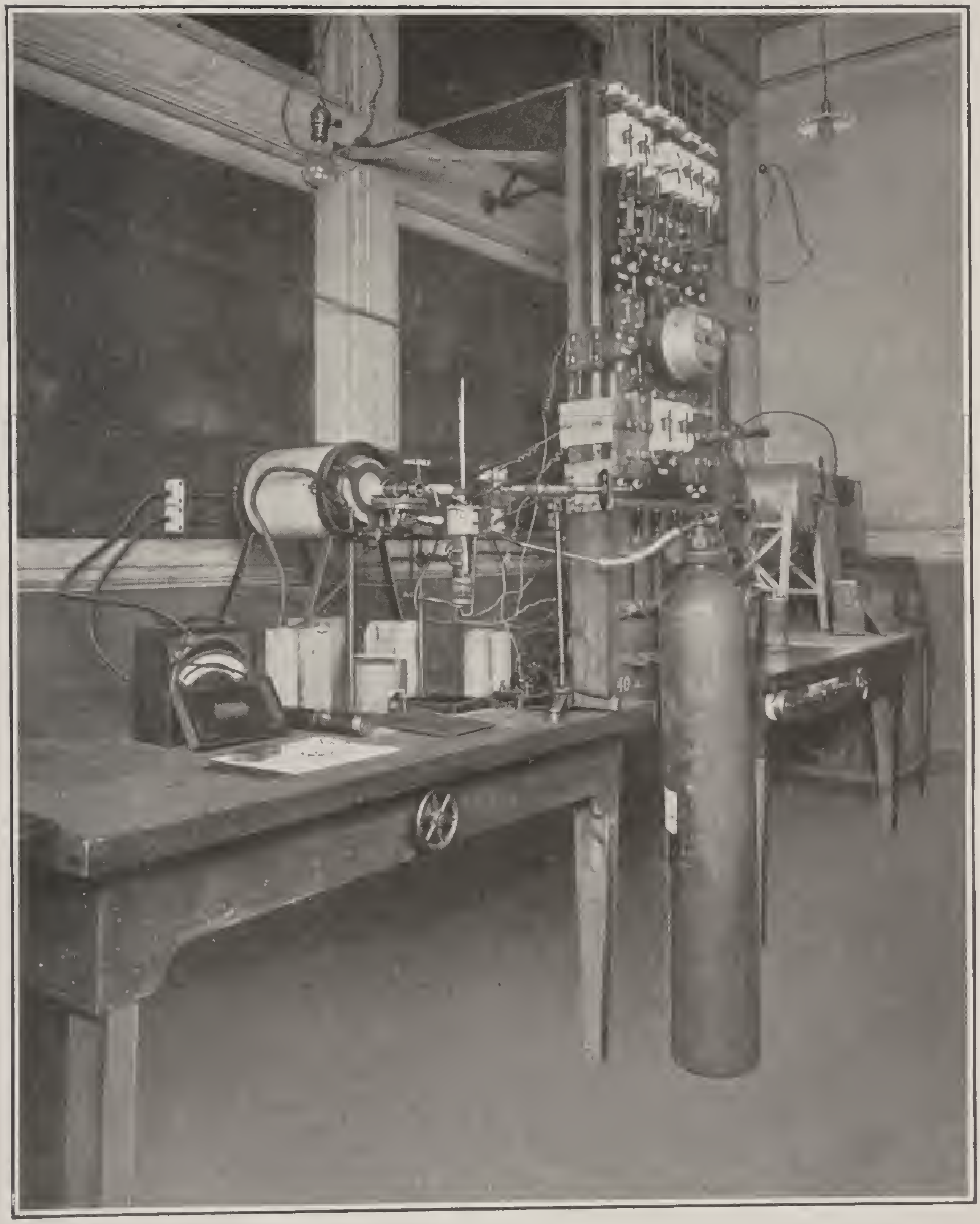


FIGURE 15.—Vertical longitudinal section through silica-tube furnace.



ARRANGEMENT OF SILICA-TUBE FURNACE AND ACCESSORIES.

wound on the inside of the alundum tube *b*, and connected in series with the platinum-foil[a] heating unit of the Heræus furnace.

This alundum heater tube was supported on two alundum rings, leaving a ½-inch air space between the heater tube and the Heræus porcelain tube. Two additional alundum tubes, *c* and *d*, were placed similarly, one at each end of the platinum-wound tube. Through the furnace, modified in this way, was passed a tube *e*, of fused silica, glazed on both sides, five-eighths inch in inside diameter and 17 inches long.

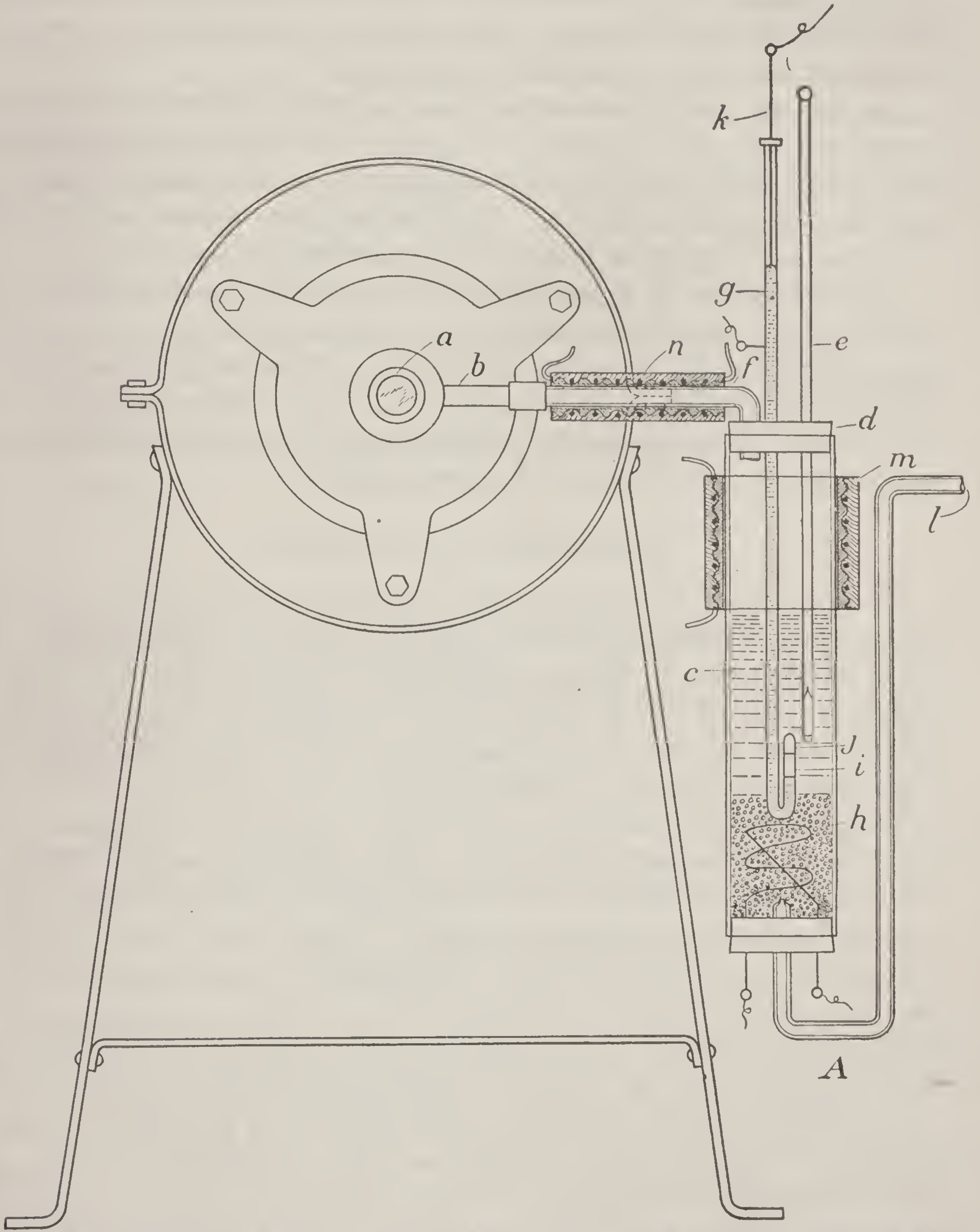


FIGURE 16.—Front elevation of silica-tube furnace and vertical section through saturator.

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[a] A nichrome heating element could as well be used for the outer heating unit.

The gas mixture entered the silica tube through the brass end piece *f*, which was specially designed to make a gas-tight connection by means of the brass bushing *g* and the asbestos packing *h*. A removable glass window, *i*, permitted removal and insertion of cones in the furnace, and observation of deformation and of temperature with the optical pyrometer. The glass was held in place between asbestos washers by a threaded brass ring.

An air bath was placed around the end piece *f* to maintain its temperature above 100° C., and thus avoid any condensation of water vapor; the exit end *g* was made sufficiently short so that its temperature should be above 100° C. The gas mixture passed out through the glass capillary tube inserted in the rubber stopper at *j*.

Alundum stops, *k*, *l*, and *m*, were placed at intervals back of the middle of the inner heater to provide a suitable background for taking readings with the optical pyrometer. These stops were made of thin-walled alundum tubes closed at one end, fitting snugly in the silica tube. To avoid rapid devitrification of the fused-silica tube, temperatures were not carried above 1,400° C.

DESCRIPTION OF "SATURATOR."

The apparatus by which various mixtures of hydrogen and water vapor were formed depended on the saturation of pure hydrogen gas with water vapor at temperatures ranging from zero to within a degree or two of the boiling point. It is shown in figure 16 with furnace connections as used, except the hot-air bath surrounding the brass end piece *a*, and the side tube *b*.

The saturator consists of a glass tube *c*, 1¾ inches in diameter and 9 inches long, closed at the ends with rubber stoppers. Through the upper stopper *d* passes the thermometer *e*, the outlet tube *f* communicating with the furnace through the brass tube *b*, and a compact adjustable vapor-pressure thermoregulator *g*, with electrical connections controlling through a relay the current in the heating coil *h*. The thermoregulator is essentially that described by Feild,[a] without the refinement of being independent of atmospheric pressure. It consists of a bent-glass tube, in the short arm of which is a small volume of liquid *i*—ether, alcohol, or water, depending on the temperature desired—resting on a mercury column. Above the liquid *i*, is a small volume of its saturated vapor *j*, the pressure of which depends only on the temperature and determines the height of the mercury column. This mercury column makes and breaks contact with the platinum wire *k*. The temperature control, with a continuous stream of hydrogen bubbling around the glass beads *h* and through the water *c* was within 0.1° to 0.2° C., which was sufficiently accurate for the purpose of the experiments. Hydrogen stored in a high-pressure cylinder was introduced at *l*, and its flow so adjusted as to give the desired current of hydrogen and water vapor through the fused-silica furnace tube. The water vapor present in the mixture of hydrogen and water vapor was prevented from condensing to liquid water on its passage to the furnace by two heating coils *m* and *n*, constructed of nichrome wire wound on alundum cores and heated, similarly to the immersed coil *h*, by a current from a low-voltage transformer operating at about 11 volts. An air-bath chamber of asbestos paper, heated by a small electric hot plate, was placed around the brass end piece *a* and the side tube *b*.

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[a] Feild, A. L., An electrical-contact vapor-pressure thermoregulator: Jour. Am. Chem. Soc., vol. 36, 1914, pp. 72–76.

Analyses were made of the mixtures of gas and water vapor issuing from the end of the silica tube. With the saturator adjusted for a mixture containing 50 per cent of each, 52 per cent hydrogen and 48 per cent water vapor were found; adjusting for an atmosphere of 89 per cent water vapor, 91 per cent was found. This degree of accuracy was considered satisfactory.

PREPARATION OF TEST PIECE.

The ash ground to 200 mesh was molded with the aid of dextrin solution into a slender triangular pyramid or Seger cone and mounted, with one side vertical, in a refractory base, according to the method previously described. On account of the small diameter of the fused-silica tube, it was possible to use only the upper half of the standard three-sixteenth by 1 inch cone adopted in the former experiments. Before the cone was placed in the furnace the dextrin binder was burned off by ignition at a red heat in a muffle.

MEASUREMENT OF TEMPERATURE.

The temperatures were read with a Wanner optical pyrometer, which was sighted on the thin alundum disk just back of the cone. A correction for absorption by the glass window was made.

The pyrometer was compared daily with the standard amyl-acetate flame, and once each week a thin piece of Kahlbaum's pure copper foil was substituted for the ash cone and its melting point read in an atmosphere of hydrogen. The results were between 1,075° and 1,090° C. (melting point, 1,083° C.).

RESULTS OF EXPERIMENTS.

Softening-temperature determinations were made with five different samples of coal ash (see Tables 1 and 2, pp. 26 and 27) containing

7 to 69 per cent of ferric oxide (Fe_2O_3), in various atmospheres ranging from pure dry hydrogen to pure water vapor. The temperature was raised at the rate of 10° to 15° C. per minute to approximately 900° C.; it was then raised 5° C. per minute until deformation began, and 2° C. per minute thereafter until the cone was down. The dry hydrogen was obtained by bubbling the hydrogen through a wash bottle of concentrated sulphuric acid. The 100 per cent water-vapor atmosphere was obtained by cutting off the hydrogen from the saturator and heating the water to boiling.

All the results obtained are given in Table 9 following, in which the softening temperature is given as a temperature interval, the lower figure being the temperature at which deformation (rounding of the apex or bending of the cone) began, and the higher figure being the temperature at which the cone bent so that the apex touched the base, or, failing to bend, softened to a lump. This latter final deformation point is meant when reference is made to the "fusion," "softening," or "deformation" point of an ash.

TABLE 9.—*Results of experiments to determine softening temperatures of ash in various mixtures of hydrogen and water vapor.*

Laboratory No.	15848	16586	15844	16243	15845
Percentage of Fe_2O_3	7.0	11.8	19.0	35.9	69.7
Percentage of H_2O in atmosphere.	Softening temperature.				
	°C.	°C.	°C.	°C.	°C.
0.0	1,322–1,360	1,390–1,402	Above 1,400	1,304–1,370	Above 1,400
0.7				1,331–1,355	
5.7	1,279–1,290	1,313–1,360	Above 4,000		Above 1,400
6.3				1,248–1,263	
7.6				1,220–1,270	
10.7			[a] 1,340		
12.4		1,279–1,313	1,155–1,271		
15.8					1,304–1,330
18.1				1,104–1,198	
19.9			1,173–1,271		
29.6				1,056–1,064	
31.2	1,179–1,271	1,227–1,251	1,173–1,263		1,227–1,322
41.7				1,048–1,062	
51.2				1,072–1,078	
57.5	1,220–1,292	1,220–1,258	1,173–1,255		1,104–1,322
77.2				[a] 1,095	
83.2				[a] 1,095	
83.9	1,173–1,295	1,241–1,249	1,227–1,263		1,206–1,350
90.3		1,227–1,251	1,206–1,295	[a] 1,095	
94.8				1,048–1,080	
100.0	1,350–1,370	1,313–1,322	Above 1,400	1,304–1,313	Above 1,402

[a] Temperature at end of deformation of cone; temperature at which deformation began not observed.

The final deformation points are plotted in figure 17, in which the abscissas represent the percentages of water vapor in the mixture, and the ordinates the temperatures of final deformation. The initial deformation points are plotted in a similar manner in figure 18.

It will be noted that both sets of curves have well-defined minima when the water vapor formed 40 to 70 per cent of the atmosphere.

The final deformation points (fig. 17), which were more accuately determined, show a fairly uniform minimum in mixtures ranging from 30 to 70 per cent water vapor. The establishment of this relation enables the formulation of a practical method of determining the lowest softening temperature of an ash in a mixture of approximately equal parts of hydrogen and water vapor. Seemingly, the gas mixture need only be controlled between limits of 30 and 70 per cent water vapor to insure the lowest softening temperature of the ash, within the unavoidable experimental deviation in determininations of this character.

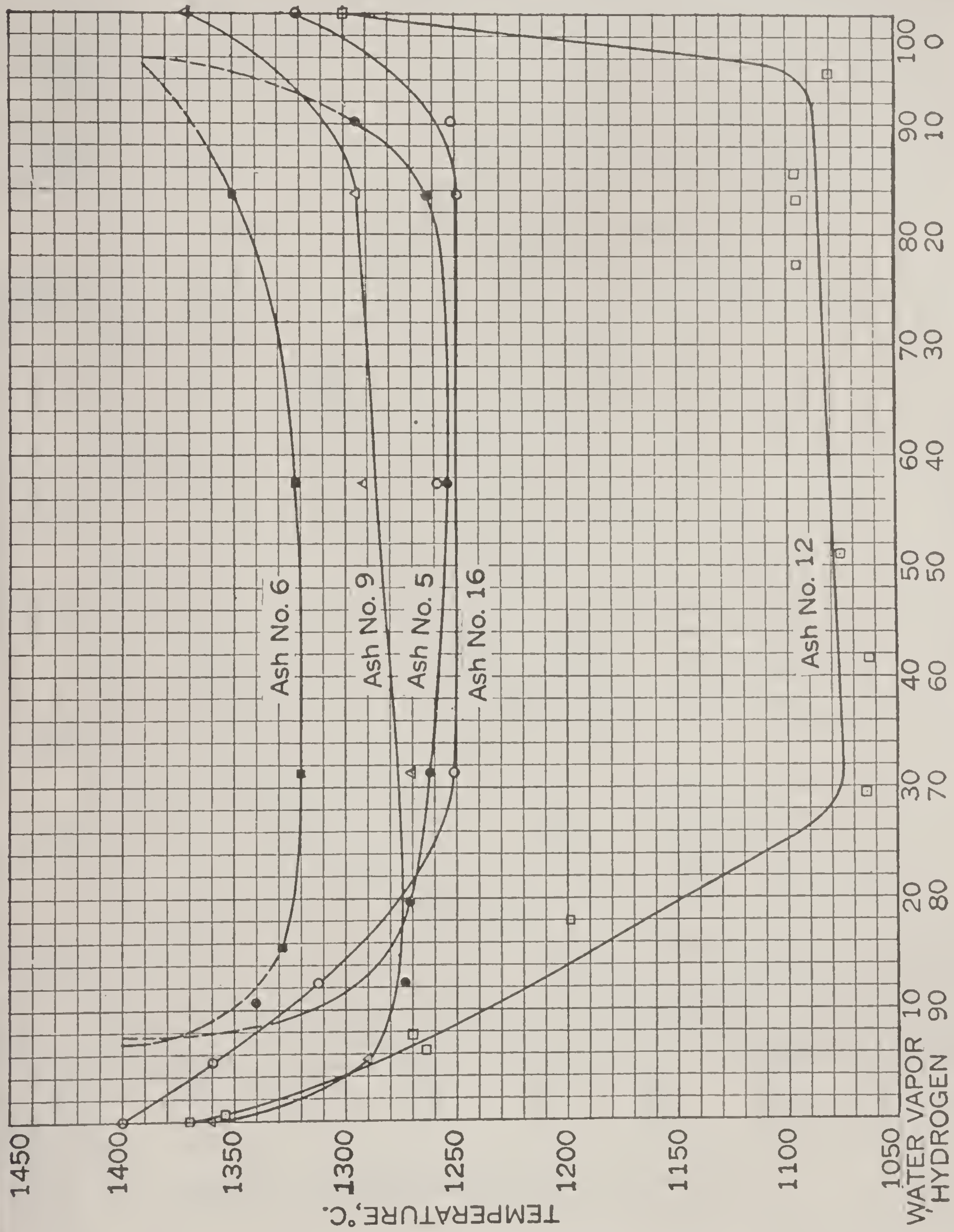

FIGURE 17.—Curves showing softening temperatures (final deformation points) of coal ash in various mixtures of hydrogen and water vapor.

The pronounced effect of viscosity on the rate of deformation of the ash cones is shown in the relative positions of the curves of initial deformation in figure 18 and the curves of final deformation in figure 17. Sample 16586 had the highest initial-deformation temperature and next to the lowest final-deformation temperature in an atmosphere containing equal parts of hydrogen and water vapor. This was due to the low viscosity of the melting eutectic formed from this ash, which contained 12.6 per cent of calcium oxide, a larger percentage of this basic oxide than was contained in any of the other samples.

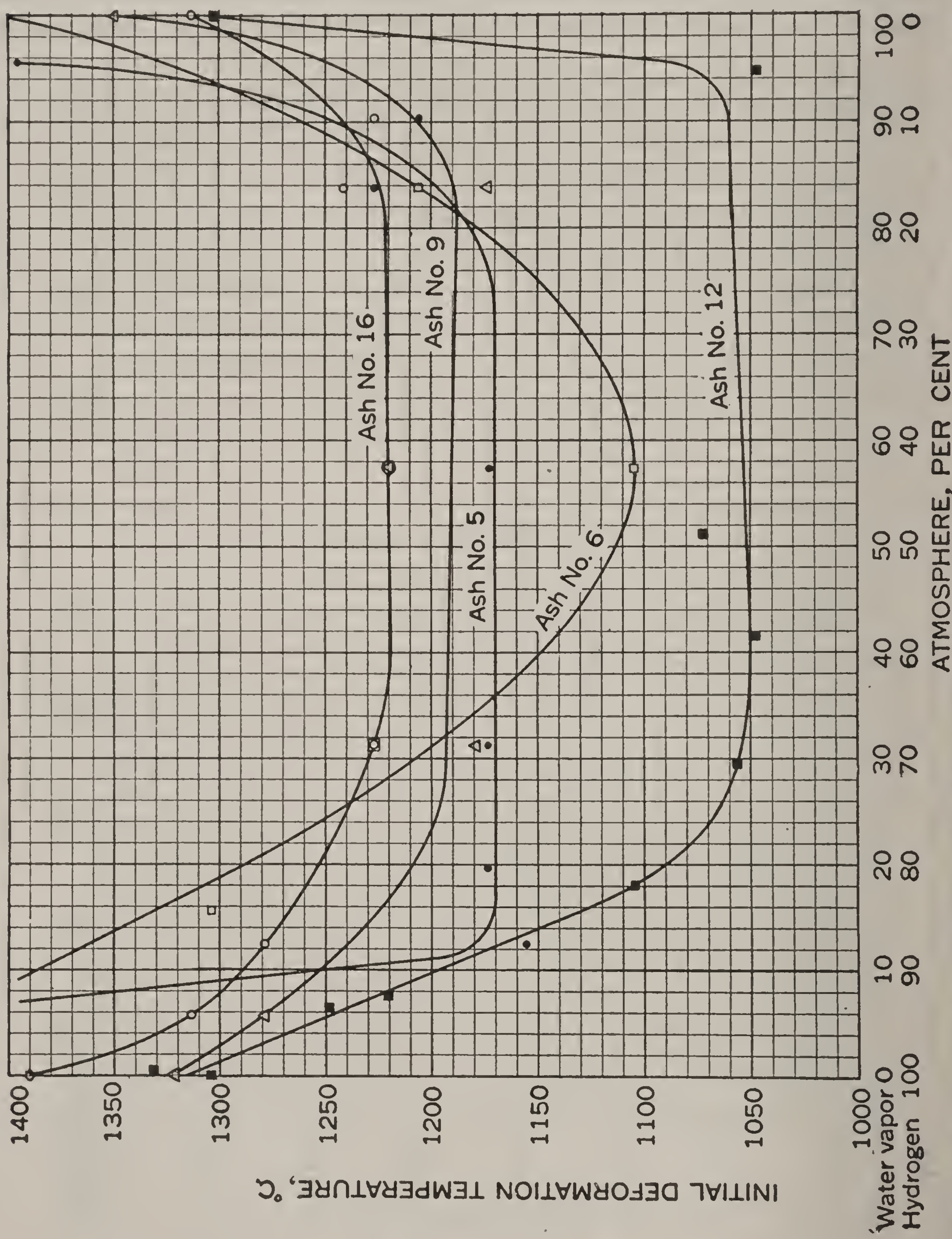

FIGURE 18.—Curves showing initial deformation temperatures of coal ash in various mixtures of hydrogen

FUSIBILITY OF COAL ASH IN MIXTURES OF CARBON MONOXIDE AND CARBON DIOXIDE.

The investigations of Kreisinger, Ovitz, and Augustine[a] have shown that, with ordinary rates of combustion, oxygen is not found in appreciable quantities in a fuel bed at distances greater than 3 or 4 inches above the grate. Samples of gas taken at a point 3 inches above the grate contained approximately 12 per cent combustible gas (principally CO) and 14 per cent carbon dioxide. In view of this predominating proportion of CO and CO_2 it is even more important to know the effect of mixtures of these two gases on the fusibility of coal ash than to know the effect of hydrogen and water vapor.

THEORETICAL CONSIDERATIONS.

Baur and Glaessner[b] found the following two series of equilibria in the system CO, CO_2, Fe, FeO, and Fe_3O_4:

$$Fe + CO_2 \rightleftarrows FeO + CO \qquad (1)$$
$$3FeO + CO_2 \rightleftarrows Fe_3O_4 + CO \qquad (2)$$

Their average values for the equilibrium constants are as follows:

Equilibrium data of Baur and Glaessner.

Reaction 1.		Reaction 2.	
Temperature of solid phase.	$\frac{pCO}{pCO_2}$	Temperature of solid phase.	$\frac{pCO}{pCO_2}$
° *C.*		° *C.*	
552	1.16	450	0.812
596	1.25	490	0.887
651	1.38	550	0.786
662	1.40	650	0.587
680	1.44	850	0.351
750	1.56	950	0.299
850	2.12		
900	2.51		

The equilibrium constants shown above are plotted in figure 19. Assuming that Fe_2O_3 reacts like Fe_3O_4 as regards reduction, it is evident that at temperatures up to 1,000° C. no reduction to metallic iron nor oxidation to Fe_3O_4 can take place in a mixture of equal parts by volume of CO and CO_2; and unless the direction of the two curves changes rather suddenly, FeO should continue to be the stable phase at higher temperatures, probably throughout the usual softening range of coal ashes. Although these conditions of equilibrium do not necessarily apply in the presence of other components as in a coal-ash fusion, they at least point out the possibility of a minimum softening point due to ferrous iron in mixtures of approximately equal parts by volume of CO and CO_2, provided this mixture is passed over the heated ash continuously, sweeping the gaseous products of the reaction out of the apparatus.

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[a] Kreisinger, Henry, Ovitz, F. K., and Augustine, C. E., Combustion in the fuel bed of hand-fired furnaces: Tech. Paper 137, Bureau of Mines, 1916, p. 39.

[b] Kreman, R. (translation by Potts, H. E.), The application of physico-chemical theory to technical processes and manufacturing methods: London, 1913, pp. 117–119.

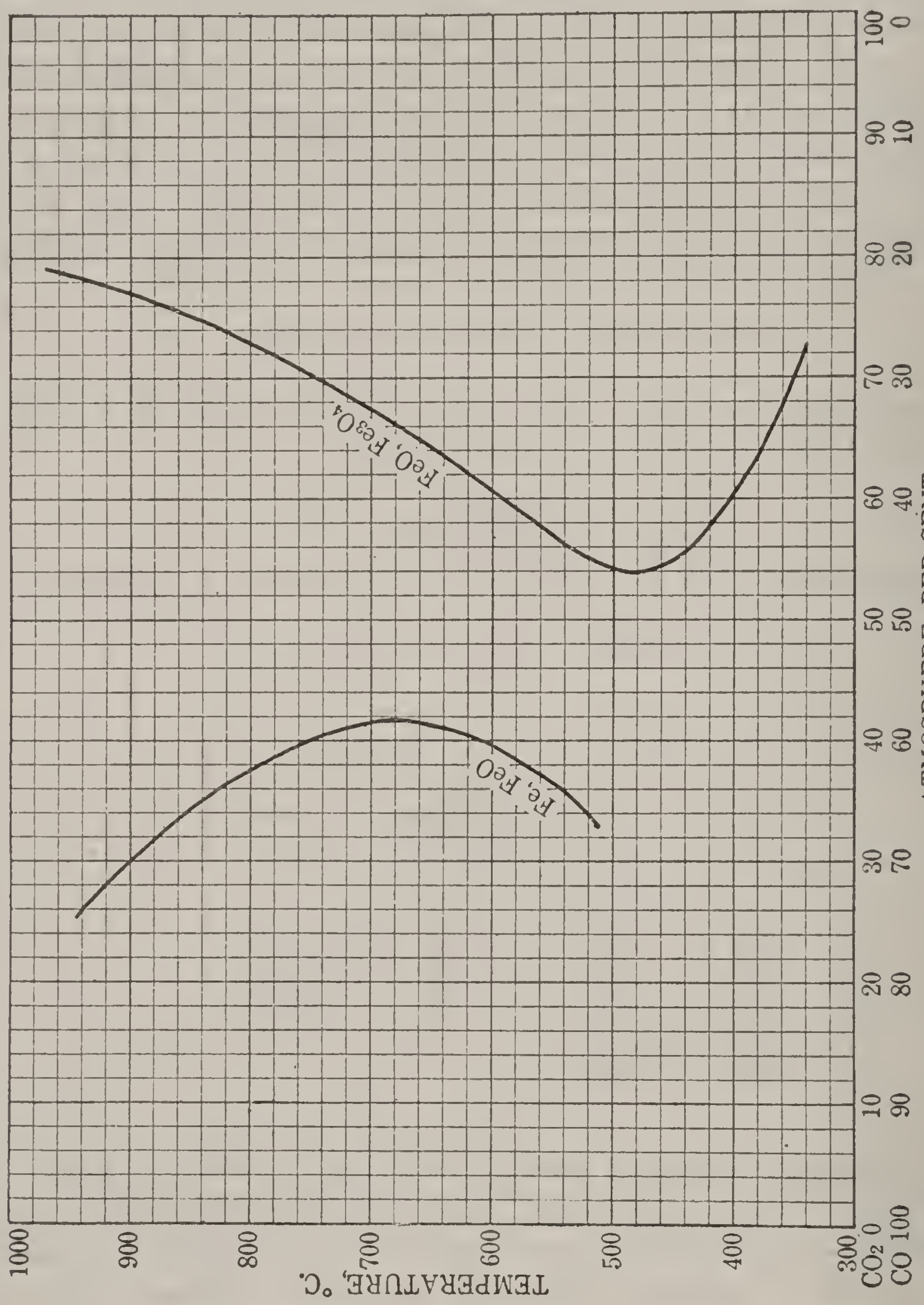

FIGURE 19.—Equilibrium diagram of the system CO, CO_2, Fe, FeO, and Fe_3O_4.

METHOD OF CONDUCTING EXPERIMENTS.

Softening-temperature determinations were made in various mixtures of CO and CO_2 with ash from the same samples that were used in the series in which atmospheres of hydrogen and water vapor were used, except sample 15,845, which had been entirely used. The same type of furnace and of silica-tube arrangement was used, the hydrogen tank and saturator being replaced with a large gasometer which contained the desired mixture of CO and CO_2. A gasometer similar to one described by Dennis[a] was found especially advantageous for this work in that a large volume of gas could be made and stored without much change in composition taking place. Instead of the usual arrangement of having the gas confined over a considerable surface of liquid, the bell was arranged to move freely up and down between two concentric cylinders; the inner one was closed at the top, thus limiting the liquid seal to the 1-inch annular space between. By using a mixture of approximately 60 per cent glycerin and 40 per cent water the solubility of CO_2 was further reduced to a minimum.

PREPARATION OF GAS MIXTURES.

The mixtures containing equal parts of CO and CO_2 were made by heating 250 grams of oxalic acid and 600 c. c. of concentrated sulphuric acid in a 1,000-c.c. side-tube distilling flask. From the flask the gas passed through a wash bottle containing concentrated sulphuric acid and then directly into the gasometer, in which a slightly diminished pressure was maintained. Air in the gasometer and connections was first removed by partly filling and then emptying the tank before the gas was stored. Mixtures containing more than 50 per cent CO_2 were made by adding more CO_2 from a tank of liquid carbon dioxide. The carbon dioxide from this latter source showed the following analysis: Carbon dioxide, 99.9 per cent; air, 0.1 per cent.

Mixtures containing less than 50 per cent CO_2 were made by interposing between the oxalic acid generator and the gasometer an absorbing train of sodium hydroxide solution for removing CO_2. CO_2 from the tank containing liquid CO_2 was then added to the CO in the gasometer. When the gasometer had been filled with the desired gas mixture the mixture was allowed to stand overnight to come to equilibrium. Samples were then taken over mercury and analyzed for carbon monoxide, carbon dioxide, and oxygen according to the Hempel method, as described in Technical Paper 63 of the Bureau of Mines.[b] Tests were made which assured the absence of any appreciable quantities of other combustible gases. In order to check any changes in the composition of the gas stored in the gasometer a sample was taken each day until the supply became exhausted. The change in composition from day to day never exceeded a few tenths of a per cent. Any small fractional percentage of oxygen found in the gas was calculated to CO_2, on the assumption that in the furnace it combined with CO, an equivalent deduction being made in the CO percentage.

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[a] Dennis, L. M., Gas analysis, 1913, pp. 25–27.

[a] Clement, J. K., Frazer, J. C. W., and Augustine, C. E., Factors governing the combustion of coal in boiler furnaces: Tech. Paper 63, Bureau of Mines, 1914, pp. 13–17, figs. 6 and 7.

RESULTS OF EXPERIMENTS.

The general procedure followed in making the first series of softening-temperature tests in CO and CO_2 was identical with that used in the tests with hydrogen and water vapor, as described on pages 57 to 60. Heating with a temperature rise of 5° C. per minute until initial deformation, and then with a temperature rise of 2° C. per minute until the cone was down gave concordant results in the hydrogen atmosphere. The same rate of heating was therefore used in the first series in which carbon monoxide was used. However, as the work proceeded it soon became apparent that wholly concordant results were not being realized; as all previous investigations on the effect of rate of heating had been made in reference to atmospheres of air or of hydrogen and water vapor, it seemed possible that a slower rate of heating might be necessary to attain uniform results in $CO–CO_2$ atmospheres. The softening temperature of ash sample 13629 was therefore determined at three different rates of heating in a mixture of approximately equal parts of CO and CO_2. As shown in the following table, when the rate of heat increase was 2° C. per minute, the ash softened at a temperature 72° C. lower than when the heat increase was 5° C. per minute.

Results of experiments to determine effect of rate of heating on softening temperature of ash.

Rate of temperature increase.	Softening temperature.	
	Initial.	Final.
	° C.	*° C.*
5° C. per minute	1,072	1,176
5° C. to initial deformation, then 2° C. to final deformation	1,048	1,152
2° C. per minute	1,022	1,104

In view of this seemingly greater lag of the reducing and softening reactions in carbon monoxide a second series of tests was made on the same ashes at a much slower rate of heating. In this series the ash was heated rapidly to a temperature of 950° C. and held at that temperature for a period of one hour; heating was then continued at a 2° C. rate until the cone was down.

The results obtained in the two series are given in Table 10 following, and in the curves of figures 20 to 23; the previously obtained curves from experiments with mixtures of hydrogen and water vapor are also plotted to the same scale to permit comparison. The results of analyses of the ashes follow:

Results of analyses of ashes represented in figures 20 to 23 and in Table 10.

Ash sample No.	SiO_2.	Al_2O_3.	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
16	37.2	25.5	11.8	1.5	12.6	1.9	1.4	0.3	5.6
5	42.2	30.6	19.0	1.2	1.3	1.0	1.3	2.9	.3
12	37.1	17.6	25.9	.7	3.2	.9	.4	1.8	2.3
9	54.8	27.0	7.0	1.3	4.3	1.7	.3	3.1	1.4

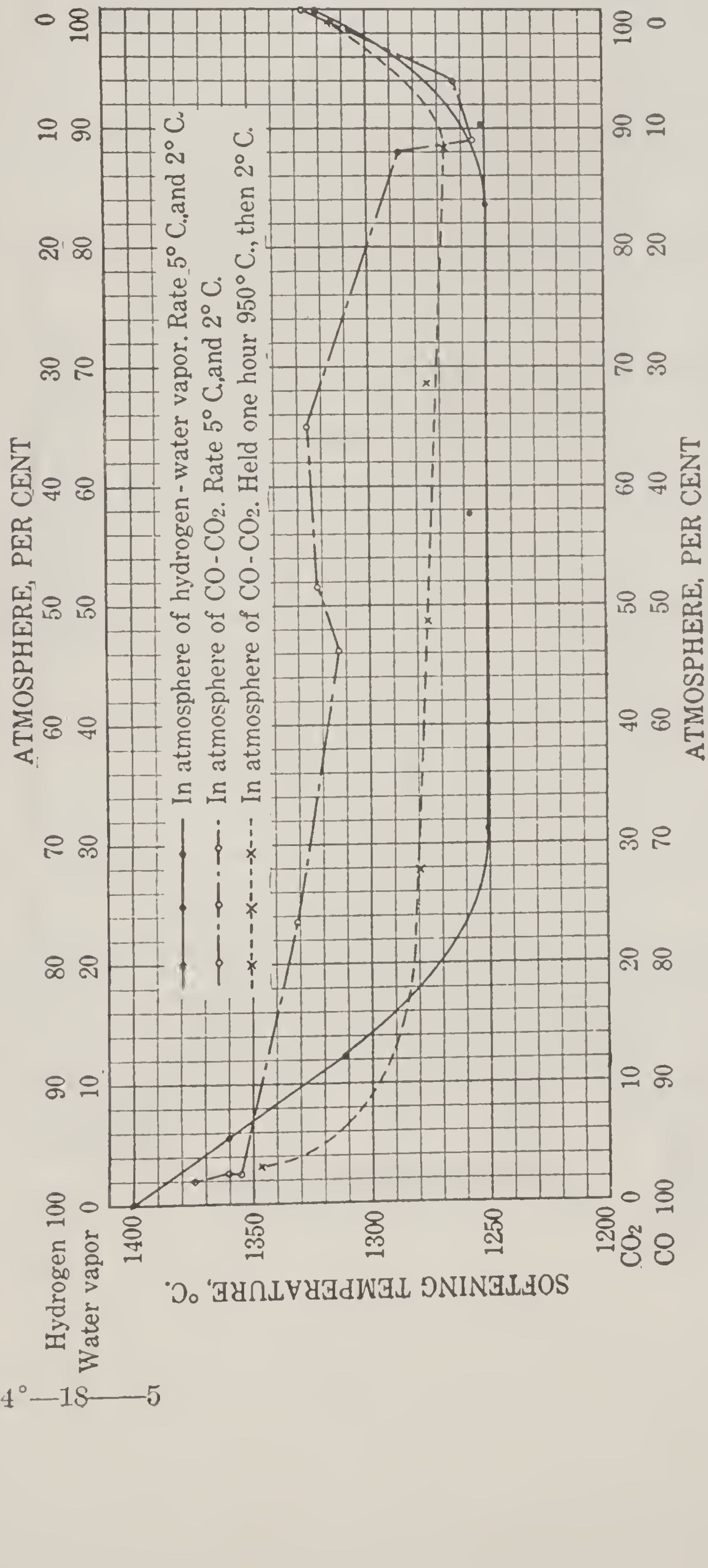

FIGURE 20.—Curves showing softening temperatures of ash sample 16 in mixtures of CO and CO_2 and of H_2 and H_2O.

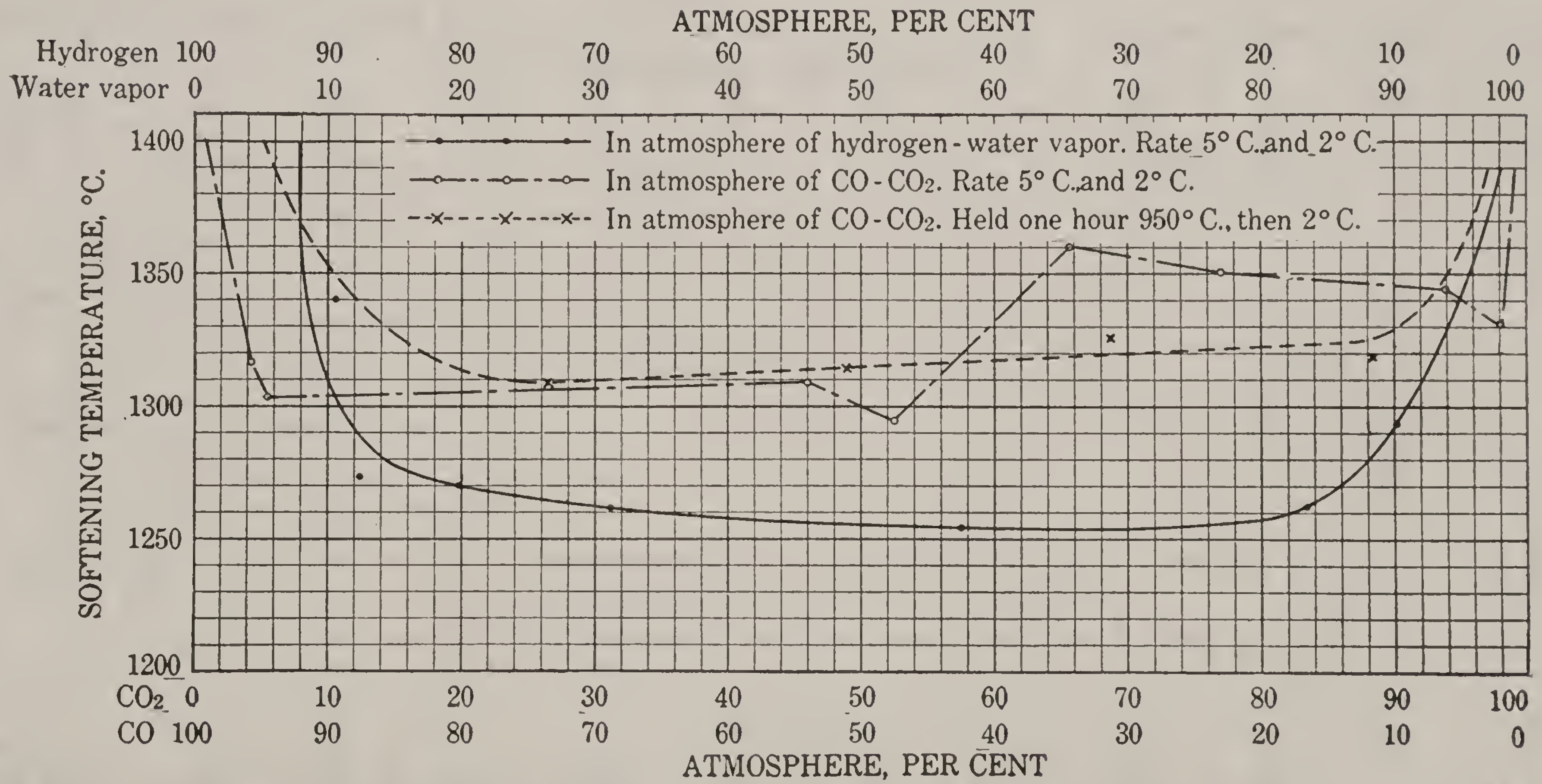

FIGURE 21.—Curves showing softening temperatures of ash sample 5 in mixtures of CO and CO_2 and of H_2 and H_2O.

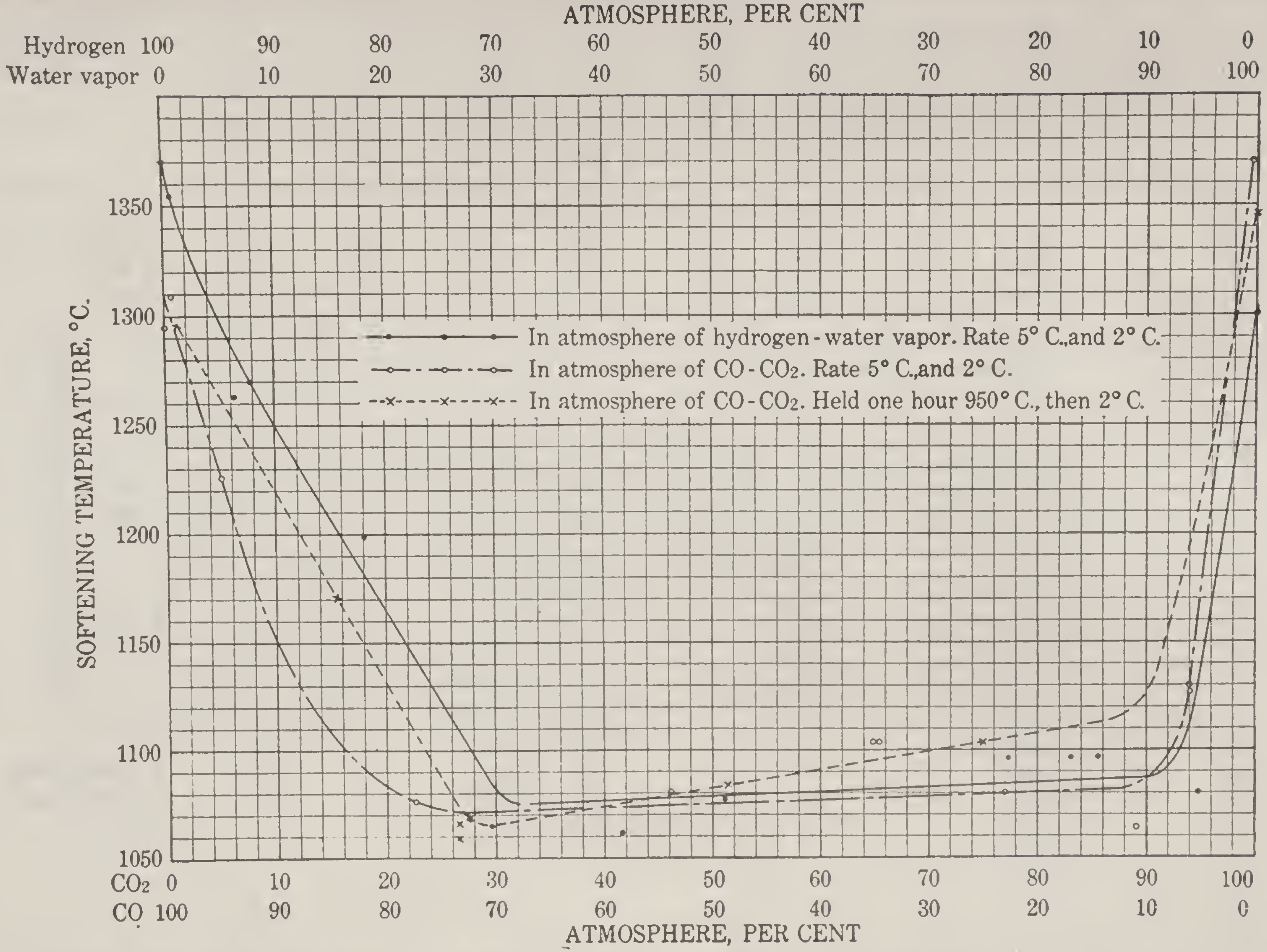

FIGURE 22.—Curves showing softening temperatures of ash sample 12 in mixtures of CO and CO_2 and of H_2 and H_2O.

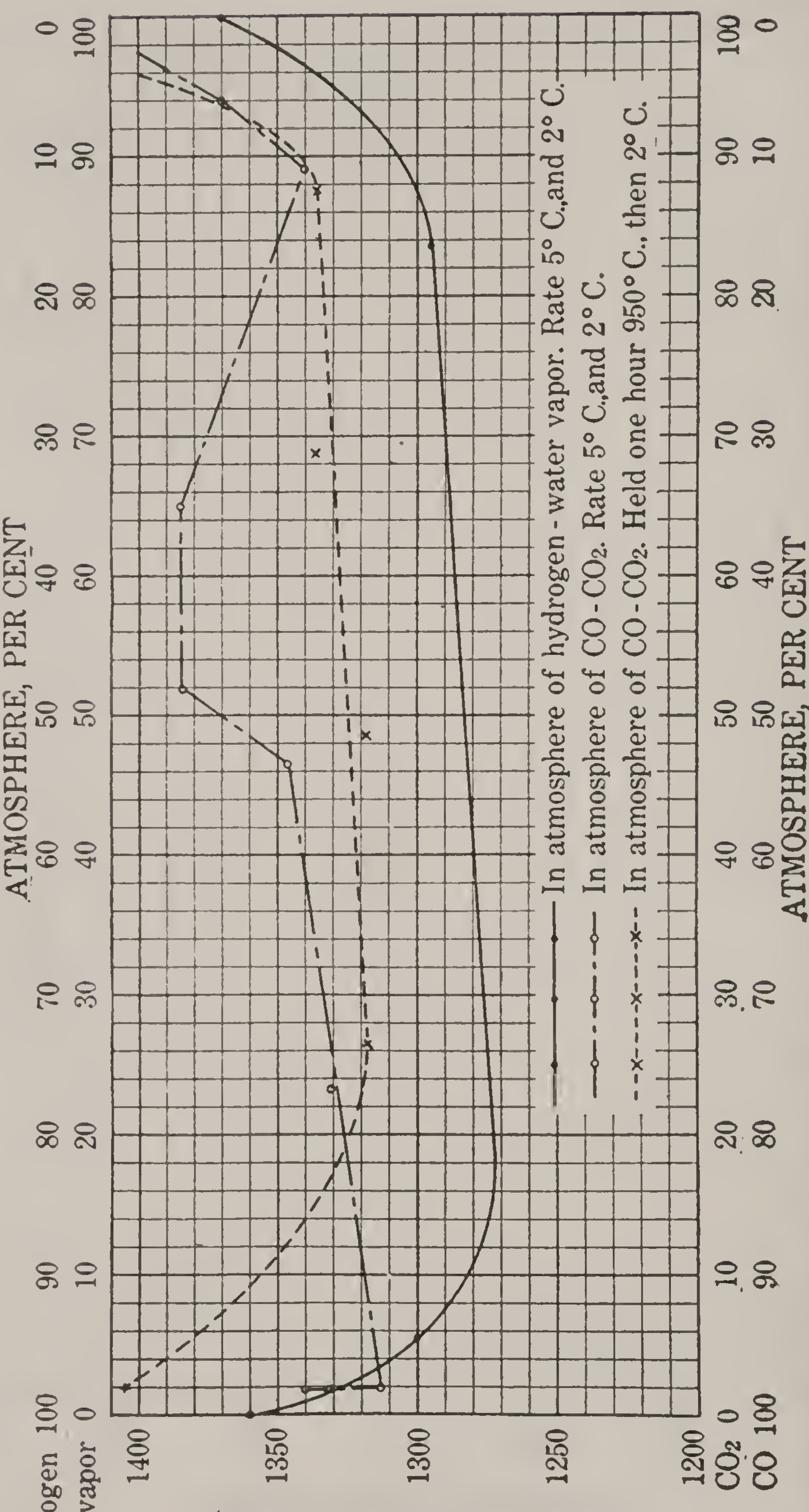

FIGURE 23.—Curves showing softening temperatures of ash sample 9 in mixtures of CO and CO_2 and of H_2 and H_2O.

TABLE 10.—*Results of tests to determine softening temperatures of certain ash samples in an atmosphere containing equal parts of CO and CO_2.*

ASH SAMPLE 9.

Composition of gas.		Softening temperature at heating rate of 5° and 2° C.[a]	Softening temperature with temperature of 900° to 950° C. for 1 hour, followed by rise of 2° C. per minute.	Magnetic properties and amount of metallic iron in fused cone.
CO.	CO_2.			
P. ct.	*P. ct.*	° C.	° *C.*	
98.2	1.8	1,227-1,331	....	Fairly magnetic; many small globules metallic Fe.
		1,276-1,340	....	Faintly magnetic; many small globules metallic Fe.
98.1	1.9	1,227-1,313	....	Faintly magnetic; a few globules of metallic Fe.
98.0	2.0	....	1,271-1,405	Fairly magnetic; many small globules metallic Fe.
76.7	23.3	1,227-1,331	....	Faintly magnetic; a few globules metallic Fe.
73.6	26.4	....	1,136-1,318	Faintly magnetic; no metallic Fe.
53.5	46.5	1,318-1,347	....	Do.
51.7	48.3	....	[a]1,237-1,318	Nonmagnetic; no metallic Fe.
48.1	51.9	1,313-1,385	....	Do.
35.2	64.8	1,350-1,385	....	Do.
31.3	68.7	....	1,175-1,337	Do.
12.7	87.3	....	1,275-1,337	Do.
11.1	88.9	1,271-1,340	....	Do.
6.0	94.0	1,304-1,370	....	Do.
.9	99.1	....	Above 1,400	Faintly magnetic; no metallic Fe.
.3	99.7	[b]Above 1,400	....	Fairly magnetic; no metallic Fe.

[a] A cone heated for 2 hours at 900° to 950° C. and then subjected to a temperature rise of 2° C. per minute thereafter started to bend at 1,174° C. and was down at 1,314° C.

[b] Just started.

ASH SAMPLE 16.

98.1	1.9	1,327-1,375	....	Strongly magnetic; many globules metallic Fe.
97.5	2.5	1,313-1,355	....	Fairly magnetic; many globules metallic Fe.
		1,322-1,360	....	Do.
97.0	3.0	....	1,318-1,347	Strongly magnetic; many globules metallic Fe.
76.7	23.3	1,252-1,331	....	Fairly magnetic; many globules metallic Fe.
72.3	27.7	....	1,167-1,279	Fairly magnetic; a few globules metallic Fe.
53.9	46.1	1,275-1,313	....	Faintly magnetic; a few globules metallic Fe.
51.7	48.3	....	[c]1,149-1,275	Faintly magnetic; very few globules metallic Fe.
48.4	51.6	1,300-1,322	....	Faintly magnetic; no metallic Fe.
35.2	64.8	1,300-1,326	....	Do.
31.5	68.5	....	1,133-1,275	Do.
12.0	88.0	1,174-1,287	....	Lost.
11.7	88.3	....	1,237-1,267	Nonmagnetic; no metallic Fe.
11.1	88.9	1,248-1,255	....	Do.
6.0	94.0	1,227-1,263	....	Fairly magnetic; no metallic Fe.
.9	99.1	....	1,233-1,314	Strongly magnetic; no metallic Fe.
.0	100.0	1,275-1,327	....	Do.

[c] A cone heated 2 hours at 900° to 950° C. and then subjected to a temperature rise of 2° C. per minute thereafter started to bend at 1,184° C. and was down at 1,275° C.

ASH SAMPLE 5.

98.2	1.8	....	1,251-[d]1,400	Fairly magnetic; many globules metallic Fe.
98.1	1.9	([e])	....	Strongly magnetic; many globules metallic Fe.
95.8	4.2	1,199-1,317	....	Do.
94.8	5.2	1,176-1,304	....	Fairly magnetic; many globules metallic Fe.
73.6	26.4	....	1,220-1,309	Faintly magnetic; no metallic Fe.
54.0	46.0	1,300-1,309	....	Nonmagnetic; no metallic Fe.
51.1	48.9	....	1,080-1,314	Do.
47.5	52.5	1,248-1,295	....	Do.
35.2	64.8	1,160-1,360	....	Faintly magnetic; no metallic Fe.
31.3	68.7	....	1,267-1,327	Do.
23.1	76.9	1,322-1,350	....	Nonmagnetic; no metallic Fe.
11.7	88.3	....	1,122-1,318	Fairly magnetic; no metallic Fe.
6.1	93.9	1,127-1,345	....	Faintly magnetic; no metallic Fe.
2.1	97.9	1,142-1,331	....	Lost.
.9	99.1	....	Above 1,400	Strongly magnetic; no metallic Fe.
.0	100.0	Above 1,400	....	Do.

[d] Above.

[e] Cone started softening at 1,243° C.; one-half melted down at 1,400° C.

TABLE 10.—*Results of tests to determine softening temperatures of certain ash samples in an atmosphere containing equal parts of CO and CO_2*—Continued.

ASH SAMPLE 12.

Composition of gas.		Softening temperature at heating rate of 5° and 2° C.[a]	Softening temperature with temperature of 900° to 950° C. for 1 hour, followed by rise of 2° C. per minute.	Magnetic properties and amount of metallic iron in fused cone.
CO.	CO_2.			
P. ct.	*P. ct.*	° C.	° *C.*	
100.0	0.0	1,270–1,295		Strongly magnetic; much metallic Fe.
99.2	.8	1,241–1,309		Do.
99.0	1.0		1,251–1,296	Strongly magnetic; many globules metallic Fe.
95.0	5.0	1,170–1,226		Strongly magnetic; much metallic Fe.
84.7	15.3		1,101–1,171	Fairly magnetic; many globules metallic Fe.
77.3	22.7	[f] 1,076		Faintly magnetic; no metallic Fe.
73.4	26.6		[a] 1,058	Nonmagnetic; No metallic Fe.
73.4	26.6		1,055–1,066	Do.
53.9	46.1	1,050–1,080		Nonmagnetic; no metallic Fe.
48.5	51.5		1,073–1,083	Do.
47.7	52.3	1,027–1,048		Faintly magnetic; no metallic Fe.
35.2	64.8	1,072–1,104		Nonmagnetic; no metallic Fe.
25.1	74.9		1,073–1,104	Do.
23.1	76.9	[f] 1,104		Lost.
		[f] 1,080		Nonmagnetic; no metallic Fe.
11.1	88.9	1,048–1,064		Fairly magnetic; no metallic Fe.
6.0	94.0	1,060–1,130		Do.
		1,048–1,127		Do.
.3	99.7	1,304–1,370		Strongly magnetic; no metallic Fe.
.0	100.0		1,300–1,347	Do.

[a] Starting point not observed.

DISCUSSION OF RESULTS.

The softening-point curves show that, except ash sample 16243, smooth curves were not obtained for the softening points in atmospheres of CO and CO_2 when the rate of increase of heating was 5° and 2° C. per minute; however, on allowing more time for reduction and softening of the ash by holding the cone at a temperature of 900° to 950° for a period of one hour and then increasing the heat 2° C. per minute until softening was complete, the seeming irregularities disappeared and curves similar in shape to those obtained in atmospheres of hydrogen and water vapor were obtained. These lower and more uniform curves show for each of the ashes tested a well-defined region of minimum softening temperature, which ranged between approximate limits of 75 and 10 per cent carbon monoxide. For most samples the minimum softening temperature in an atmosphere of carbon monoxide and carbon dioxide was somewhat higher than the softening temperature in an atmosphere of hydrogen and water vapor, the greatest difference being 65° C. for sample 15844.

The general similarity in shape of the H_2–H_2O and the CO–CO_2 curves would indicate that the variation in the fusibility of a given ash is determined principally by the state of oxidation of its iron content. The stable phase of iron is in turn determined by the temperature, by the relative concentrations of oxidizing and reducing gases, and, in part, by the other components of the ash. In a general way the effect as regards reduction would be similar for carbon monoxide and hydrogen; likewise the oxidizing effects of air, of carbon dioxide, and of water vapor on the iron component are similar enough to give maximum softening temperatures of the same order for a given ash, as shown in Table 11 following:

TABLE 11.—*Results of tests to determine comparative effects on softening temperatures of ash of atmospheres of air, of water vapor, of carbon dioxide, of carbon monoxide, of equal parts of hydrogen and water vapor, and of equal parts of carbon monoxide and carbon dioxide.*

Ash sample No.	Per cent of Fe_2O_3.	Softening temperature, ° C.						
		Oxidizing atmosphere.			Reducing atmosphere.		Atmosphere of equal parts of H_2 and H_2O.	Atmosphere of equal parts of CO and CO_2.
		Air.	Water vapor.	CO_2.	H_2.	CO.		
16243	35.9	1,335	1,313	1,347	1,370	1,295	1,080	1,085
15844	19.0	1,490	(a)	(a)	(a)	(a)	1,255	1,315
16586	11.8	1,322	1,322	1,327	1,402	1,375	1,250	1,275
15848	7.0	1,425	1,370	(a)	1,360	1,331	1,283	1,325

[a] Above 1,400.

The sharp rise of the softening-point curves at both the oxidizing and reducing end confirms the reasons previously advanced for the large variations in results obtained in different furnaces. It is evident that no concordant results can be obtained in oxidizing atmospheres containing small proportions of reducing gas or similarly in essentially reducing atmospheres with small admixtures of oxidizing gases, as in these regions the curves have a sharp inclination.

RELATION OF SOFTENING TEMPERATURES IN ATMOSPHERES OF HYDROGEN AND WATER VAPOR, AND OF CARBON MONOXIDE AND CARBON DIOXIDE TO CLINKER FORMATION IN FURNACES.

The general form of the softening-point curves suggests a choice of three types of atmosphere in which uniform softening points may be determined, namely, (1) a strongly reducing atmosphere of 100 per cent hydrogen or carbon monoxide; (2) a weakly reducing atmosphere of approximately 50 per cent hydrogen and 50 per cent water vapor, or of 50 per cent carbon monoxide and 50 per cent carbon dioxide; (3) an oxidizing atmosphere of 100 per cent water vapor, carbon dioxide, or air.

The atmosphere to be selected for a standard softening-temperature test should be one that produces a slag with the iron in the same state of oxidation as is usually found in fuel-bed clinkers. Determinations were therefore made of the relative percentages of ferrous, ferric, and metallic iron in the glassy parts of furnace clinkers and in the ash cones fused in the atmospheres mentioned above.

FERROUS, FERRIC, AND METALLIC IRON IN FUSED ASH CONES AND CLINKER SLAGS.

Fourteen to twenty ash cones, whose weight aggregated 2 or 3 grams, were placed in a platinum boat and heated to softening temperatures in the silica tube of the furnace at the different rates of heating used in the previous tests. In the tests with an atmosphere of hydrogen and water vapor the boat and its contents were held at this temperature for 30 minutes to permit the softening reactions to approach equilibrium. In the tests with an atmosphere of CO and CO_2 heating was stopped as soon as the softening point was reached. As in the softening-temperature determinations the cones were heated to redness previous to the tests to oxidize and remove the organic binders. At the end of the heating period the electric current was cut off and the furnace allowed to cool rapidly, without interrupting the gas stream. To avoid any possibility of change during cooling it would have been better to have quenched the fusion suddenly. However, the viscosity of the ash melted was so great that little change was probable at the rate of cooling used. The fused material appeared glassy.

As the particular ash samples used in the softening-temperature tests and in the fusions with hydrogen and water vapor became exhausted before the fusions in atmospheres of CO and CO_2 were made, it was necessary to use for the latter fusion tests other samples of ash from coal from the same mines. The new samples showed a slightly different total iron content and a softening temperature that deviated approximately 40° C. from that of the original samples. However, they served as well for use in determining the relative reducing effect of the different atmospheres on the ferric oxide.

After the furnace had cooled, the fused cones were removed from the boat and were pulverized, first by impact in a diamond mortar of hardened steel and then in an agate mortar, to 100 mesh and analyzed for ferrous, ferric, and metallic iron by the method of Mathesius.[a] The results, as given in Table 12 following, show well-defined differences in the state of oxidation of the iron content of the fused ash samples in the various atmospheres. In air, water vapor, and carbon dioxide 67 to 88 per cent of the ferric oxide remained as ferric iron; in hydrogen and in carbon monoxide, 49 to 78 per cent of the iron was reduced to the metallic form, the larger percentage being found in the cones fused in hydrogen atmosphere; in the cones fused in an atmosphere of equal parts of hydrogen and water and in the cones fused in an atmosphere of equal parts of CO and CO_2 little or no metallic iron was found, the dominant form being ferrous iron, the proportion of which ranged from 78 to 94 per cent.

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[a] Mathesius, Ludwig, Untersuchungen über die Reduzierbarkeit von Eisenerzen in strömenden Gasen: Stahl und Eisen, Bd. 34, 1914, pp. 866–867.

TABLE 12.—*Results of analyses of ash cones fused in various atmospheres and of furnace clinkers.*

Sample No.	Description of sample.	Atmosphere.	Rate of temperature rise per minute.	Maximum temperature of fusion, ° C.	Total Fe.	Percentage of total Fe occurring as—			Character of slag.
						Metallic iron.	Ferrous iron.	Ferric iron.	
GROUP I. FERRIC IRON PREDOMINATING.									
					P. ct.				
20155.	Ash	CO_2	5° and 2° C.[a]	1,405	6.7	None.	12	88	Glassy, black; slightly magnetic.
20155.	do	do	(b)	1,405	6.8	None.	24	76	Glassy, black; nonmagnetic.
20242.	do	do	5° and 2° C.[a]	1,368	21.4	None.	28	72	Black, glassy; highly magnetic.
20242.	do	do	(b)	1,368	21.3	None.	33	67	Do.
16243.	do	H_2O vapor.	5° and 2° C.[a]	1,300	27.9	Trace.	28	72	Glassy, black, highly magnetic.
13629.	do	Air	do	1,400	15.5	Trace.	30	70	Dark red to almost black, glassy; highly magnetic.
GROUP II. METALLIC IRON PREDOMINATING.									
20155.	Ash	96CO;4CO_2	5° and 2° C.[a]	1,337	6.6	49	41	10	Dark gray; magnetic.
20155.	do	do	(b)	1,405	6.2	55	32	13	Do.
20242	do	99CO;1CO_2	5° and 2° C.[a]	1,309	20.7	60	36	4	Dark gray; highly magnetic.
20242	do	97CO;3CO_2	(b)	1,309	21.9	62	38	Trace	Do.
16243.	do	H_2	5° and 2° C.[a]	1,370	28.6	78	21	1	Black; metallic luster.
GROUP III. FERROUS IRON PREDOMINATING.									
20155	Ash	48CO;52CO_2	5° and 2° C.[a]	1,368	6.6	None.	81	19	Dark gray; slightly magnetic.
20155	do	do	(b)	1,405	5.7	None.	78	22	Do.
20242.	do	47 CO; 53 CO_2.	5° and 2° C.[a]	1,080	20.9	4	88	8	Dark, glassy; slightly magnetic.
20242.	do	do	(b)	1,080	20.9	6	79	15	Do.
20242.	do	48 CO; 52 CO_2.	5° and 2° C.[a]	1,368	20.4	7	87	6	Do.
20242.	do	do	(b)	1,370	19.8	8	84	8	Do.
16243.	do	50 H_2; 50 H_2O.	5° and 2° C.[a]	1,080	27.4	Trace.	82	18	Black, glassy; nonmagnetic.
15848.	do	do	do	1,280	4.7	6	94	Trace	Gray, glassy; nonmagnetic.
20137.	Clinker from a hand-fired furnace.				14.5	12	79	9	Dark, glassy; contained some magnetic pieces.
20145.	Clinker from boiler furnace.				12.7	6	75	19	Brown and black, glassy; contained some magnetic pieces.
20452.	do				8.9	9	75	16	Do.

[a] Cone heated with a temperature rise of 5° C. per minute until initial deformation when the rate of increase was 2° C. per minute until final deformation.

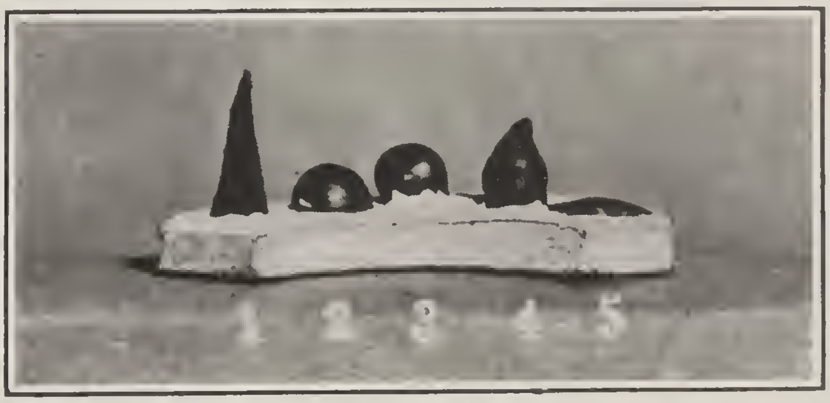
[b] Cone heated for 1 hour at 950° C., when it was subjected to a temperature rise of 2° C. per hour.

Together with the results of analyses of fused ash cones are given the results of analyses of the fused and glassy parts of several clinkers produced under working conditions in the fuel beds of different boiler furnaces operating with Pittsburgh coal. The coal, however, came from different mines. Clinker 20137 was obtained from the grate of a small hand-fired experimental furnace used for combustion investigations. Clinkers 20145 and 20452 were kindly furnished by Mr. A. A. Straub, steam engineer of the Duquesne Light Co., of Pittsburgh, Pa. These clinkers were dumped from the fuel bed while hot and were quenched with water. The clinkers were of the particularly troublesome taffylike variety that flow over the grate bars and shut off the air supply for combustion. Plate III, *A*, represents a photograph of clinker 20452. The edge at the top of the plate is the side that adhered to the corrugated grate of the new model type D Roney stoker. The matrix of this clinker was a black glass which could readily be chipped out in quantity for analysis without including any coke or unfused material. At the contact surface of coke and slag, particles of metallic iron could be identified. A microscopic examination, made by Mr. A. A. Klein, of the Bureau of Standards, of the powdered slag from this clinker and from the others showed the material to be essentially a glass with small quantities of sillimanite (Al_2O_3,SiO_2) and of an opaque iron mineral, probably magnetite, as the only crystalline phases present.

The results of analyses of the fused ash cones, and of the three fuel-bed clinkers given in Table 12 show that of the different atmospheres employed in the laboratory furnace, the atmosphere containing equal parts of hydrogen and water vapor and that containing equal parts of carbon monoxide and carbon dioxide produced slags that corresponded most closely in the state of oxidation of their iron content to the glassy parts of the furnace clinkers. In both the fused cones and the clinkers approximately 80 per cent of the iron appeared as ferrous iron, the form in which it imparts the maximum fluxing action on the silicate mixture.

In view of the results presented above, the conclusion is warranted that the minimum softening temperature of a coal ash as determined in either the $CO-CO_2$ or the H_2-H_2O atmosphere is more representative of the temperature of clinker formation under furnace conditions than the somewhat higher results obtained in strongly reducing atmospheres of hydrogen or carbon monoxide, on the one hand, and oxidizing atmospheres of air, water vapor, or carbon dioxide, on the other.

A. CLINKER NO. 20452 FROM GRATE OF RONEY STOKER.



B. TYPICAL FORMS OF CONES FUSED IN THE NO. 3 MELTER'S FURNACE.

RELATIVE MERITS OF $CO-CO_2$ AND OF H_2-H_2O ATMOSPHERES AS STANDARD ATMOSPHERES FOR FUSION TESTS.

The results of the experiments with the four ashes that have been investigated indicate that the effects of hydrogen and of carbon monoxide as a reducing agent are closely similar. The principal effect of either gas mixture is to cause the iron component to appear in the slag in the ferrous state. There are, however, certain minor reactions involving the formation of sulphides and carbides that proceed differently in the two atmospheres. For example, all the ash cones fused in an atmosphere containing equal parts of CO and CO_2 and in an atmosphere of 100 per cent CO, contained ferrous sulphide and some of the fused cones contained carbides; the sulphides were formed by reduction of sulphates in the ash. In the fusions made in an H_2-H_2O atmosphere special evidence of sulphide formations was not noted. Obviously carbides could not be formed in the latter atmosphere. These minor reactions and perhaps a slower reaction velocity of reduction may be the cause of the somewhat higher softening temperatures obtained in the $CO-CO_2$ mixtures.

In order to obtain further data on the relation between the minimum softening temperatures of various ashes, as determined in an atmosphere containing equal parts of H_2 and H_2O, in an atmosphere containing equal parts of CO and CO_2, and in air, many samples of coal ashes having various contents of iron oxide were tested in these atmospheres.

The results are given in Table 13, following, and are plotted in the order of increasing iron oxide content in figures 24 and 25. Table 13 and figure 24 also present a series of results obtained in gas furnace No. 3. These values are discussed in a subsequent part of this report.

TABLE 13.—*Results of tests to determine softening temperatures of coal ashes containing various percentages of ferric oxide in various atmospheres.*

[Each cone $\frac{1}{16}$ by $\frac{1}{2}$ inch, mounted in a vertical position in quartz tube in furnace shown in figure 17, except certain tests in a gas furnace; rate of increase of heating for tests in atmosphere of equal parts of H_2 and H_2O, 5° C. per minute until initial deformation, then 2° C. per minute; for tests in atmosphere of equal parts of CO and CO_2 rate of increase of heating was 2° C. per minute.]

Ash sample No.	Fe_2O_3 in ash.	Softening temperature in atmosphere of:				Differences.				Softening interval.				Magnetism of cones fused in atmosphere of equal parts of CO and CO_2.
		Equal parts of H_2 and H_2O.	Equal parts of CO and CO_2.	Gas furnace.	Air.	Atmosphere of H_2 and H_2O and atmosphere of CO and CO_2.	Atmosphere of H_2 and H_2O and air.	Atmosphere of CO and CO_2.	Atmosphere of gas furnace and atmosphere of CO and CO_2.	H_2–H_2O atmosphere.	CO–CO_2 atmosphere.	Atmosphere of gas furnace.	Air.	
	P. ct.	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*	
96	5.0	1,263	1,254	1,266	1,278	− 9	15	24	− 12	36	4	5	5	Not magnetic.
92	5.9	1,345	1,405	1,361	1,463	60	118	58	+ 44	125	134	21	8	Do.
97	6.5	1,313	1,292	1,236	1,345	− 21	32	53	+ 56	42	70	36	15	Slightly magnetic.
75	6.8	a1,327	1,405	1,469	1,520	78	193	115	− 64	36	122	38	64	Not magnetic.
9	7.0	1,271	1,330	1,369	1,425	59	154	95	− 39	92	177	54	55	Do.
77	7.0	1,300	1,360	1,401	1,470	60	170	110	− 41	77	89	113	60	Do.
95	7.4	1,227	1,288	1,325	1,370	61	143	82	− 37	54	107	148	20	Do.
13	7.8	1,209	1,345	1,262	1,500	136	291	155	+ 83	51	181	158	60	Do.
85	8.0	a1,228	1,233	1,240	1,320	5	92	87	− 7	37	83	84	20	Do.
73	8.2	1,255	1,405	1.358	1,450	150	195	45	+ 47	49	214	176	120	Do.
76	8.4	1,205	1,261	1,250	1,400	56	195	139	+ 11	56	91	21	120	Do.
71	9.0	1,179	1,320	1,298	1,465	141	286	145	+ 22	75	193	90	65	Do.
18	9.0	1,187	1,335	1,277	1,400	148	213	65	+ 58	52	186	121	60	Do.
74	9.2	1,227	1,370	1,327	1,405	143	178	35	+ 43	35	230	122	75	Do.
15	9.4	1,216	1,325	1,310	1,407	109	191	82	+ 15	31	134	138	67	Do.
78	9.5	1,212	1,271	1,282	1,405	59	193	134	− 11	38	149	48	55	Do.
69	9.5	1,161	1,197	1,166	1,350	36	189	153	+ 31	6	85	36	98	Do.
72	9.8	1,173	1,370	1,304	1,426	197	253	56	+ 66	69	243	75	96	Do.
17	10.1	a1,198	1,201	1,218	1,343	3	145	142	− 17	41	27	62	63	Do.
64	10.3	a1,291	1,310	1,240	1,474	19	183	164	+ 70	100	120	32	65	Do.
68	10.5	1,173	1.201	1,229	1,350	28	177	149	− 28	25	51	73	97	Do.
99	11.3	b1,201	1,215	1,197	1,299	14	98	84	+ 18	10	93	41	10	Do.
16	11.8	a1,251	1,261	1,261	1,322	10	71	61	0	24	49	53	82	Do.
82	11.9	1,181	1,310	1,325	1,456	129	275	146	− 15	63	177	64	66	Do.
79	12.4	1,149	1,310	1,336	1,414	161	265	104	− 26	48	170	21	46	Do.
89	13.3	a1,155	1,292	1,325	1,369	137	214	77	− 33	75	222	143	12	Do.
65	13.6	a1,165	1,190	1,166	1,409	25	244	219	+ 24	24	50	36	27	Do.
81	15.0	1,143	1,271	1,304	1,418	128	275	147	− 33	42	131	96	38	Do.
90	15.4	a1,170	1,212	1,177	1,324	42	154	112	+ 35	16	42	47	21	Do.
70	16.6	1,148	1,335	1,176	1,410	187	262	75	+159	44	182	31	28	Do.

8	18.0	1,205	1,365	1,320	1,450	160	245	85	+ 45	41	273	5	10	Slightly magnetic.
80	18.1	1,167	1,320	1,226	1,430	153	263	110	+ 94	66	170	32	30	Do.
67	18.3	[a]1,185	1,360	1,316	1,450	175	265	90	+ 44	50	258	69	10	Not magnetic.
100	18.9	[b]1,196	1,212	1,172	1,377	16	181	165	+ 40	31	142	26	20	Do.
5	19.0	1,255	1,314	1,340	1,490	59	235	176	− 26	82	181	54	50	Do.
66	19.5	[a]1,188	1,212	1,218	1,340	24	152	128	− 6	23	22	62	10	Do.
83	20.0	[a]1,116	1,190	1,145	1,368	74	252	178	+ 45	36	140	41	31	Slightly magnetic.
11	20.4	[a]1,132	1,143	1,156	1,400	11	268	257	− 13	47	62	26	20	Not magnetic.
10	22.4	[a]1,137	1,153	1,187	1,310	16	173	157	− 34	10	45	42	40	Do.
98	23.5	1,105	1,104	1,166	1,322	− 1	217	218	− 62	55	82	87	13	Do.
93	24.9	1,127	1,170	1,198	1,370	43	243	200	− 28	23	78	84	20	Do.
84	28.8	1,108	1,092	1,120	1,276	− 16	168	184	− 28	28	7	41	26	Do.
94	29.7	1,185	1,268	1,272	1,345	83	160	77	− 4	4	87	64	70	Slightly magnetic.
87	31.5	[a]1,119	1,092	1,116	1,305	− 27	186	213	− 24	33	38	12	26	Not magnetic.
91	34.6	1,263	1,292	1,341	1,497	29	234	205	− 49	90	133	211	97	Slightly magnetic.
12	35.9	[a]1,060	1,080	1,064	1,335	20	275	255	+ 16	12	30	10	65	Do.
86	41.9	1,071	1,077	1,076	1,358	6	287	281	+ 1	6	51	27	37	Not magnetic.
88	61.7	[a]1,113	1,130	1,125	1,452	17	339	322	+ 5	0	18	15	152	Slightly magnetic.
Average	15.9	1,193	1,259	1,249	1,393	66	200	134	10	44	117	63	49	

[a] Cone $\frac{3}{16}$ by 1 inch, mounted vertically in molybdenum furnace shown in figure 29.

[b] Atmosphere of 60 per cent H_2 and 40 per cent H_2O used instead of 50 per cent of each.

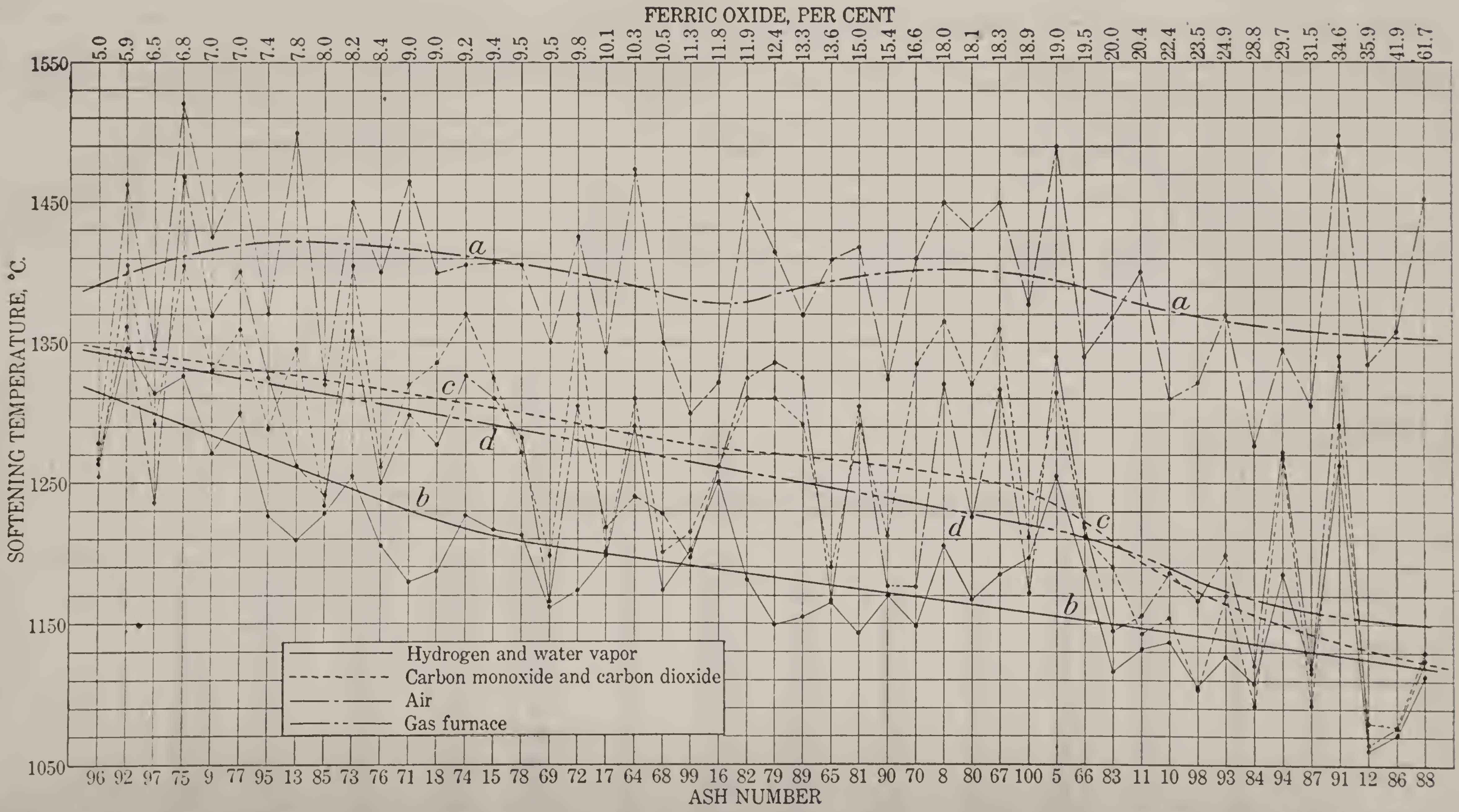


FIGURE 24.—Curves showing softening temperatures of coal ash in atmospheres of equal parts of H_2 and H_2O, equal parts of CO and CO_2, gas furnace No. 3, and air, with especial reference to the percentage of Fe_2O_3 in the ash. *a*, Average curve showing relation between softening temperature in air and percentage of Fe_2O_3 in ash; *b*, average curve showing relation between softening temperature in atmosphere of equal parts of H_2 and H_2O and percentage of Fe_2O_3 in ash; *c*, average curve showing relation between softening temperature in atmosphere of equal parts of CO and CO_2 and percentage of Fe_2O_3 in ash; *d*, average curve showing relation between softening temperature in gas furnace No. 3 and percentage of Fe_2O_3 in ash.

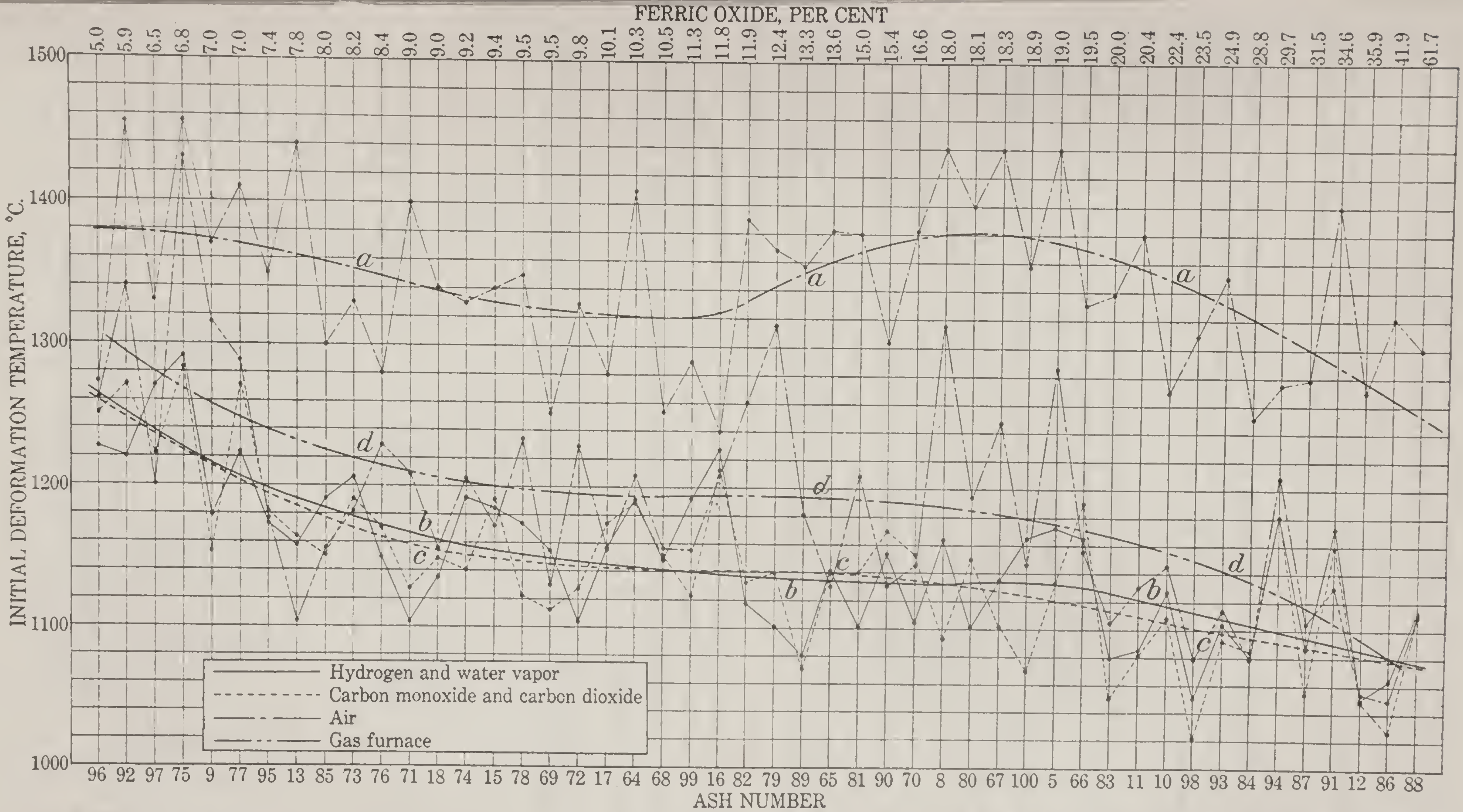


FIGURE 25.—Curves showing initial deformation points of coal ash in atmospheres of equal parts of H_2 and H_2O, equal parts of CO and CO_2, gas furnace No. 3, and air, with especial reference to the percentage of Fe_2O_3 in the ash. *a*, Average curve showing relation between initial deformation point in air and percentage of Fe_2O_3 in ash; *b*, average curve showing relation between initial deformation point in atmosphere of equal parts of H_2 and H_2O and percentage of Fe_2O_3 in ash; *c*, average curve showing relation between initial deformation point in atmosphere of equal parts of CO and CO_2 and percentage of Fe_2O_3 in ash; *d*, average curve showing relation between initial deformation point in gas furnace No. 3 and percentage of Fe_2O_3 in ash.

It will be seen from figure 25 that softening of the ash began in reducing atmospheres of H_2 and H_2O and of CO and CO_2 at practically the same temperatures, whereas in oxidizing atmospheres of air softening did not take place until a considerably higher temperature had been reached, the average difference varying from 100° to 250° C.

Many ashes, however, showed distinct differences in the temperature at which complete deformation of the cones took place in the atmospheres of H_2 and H_2O and of CO and CO_2, as shown in figure 24.

The curves representing average softening temperatures show that the differences gradually increased with an increase of Fe_2O_3—from 30° C. for an ash with 5 per cent Fe_2O_3 to 90° C. for an ash with 9 per cent Fe_2O_3. The average difference remained constant until 19 per cent Fe_2O_3 was exceeded when it rapidly decreased to 20° C. and less for ashes having 25 per cent or more of ferric oxide.

Individual samples showed differences both above and below the average curves. The largest observed difference was 197° C. The mean difference for the 48 samples tested was 66° C.

Seemingly, these differences in softening intervals of ash samples in atmospheres of H_2 and H_2O and of CO and CO_2 have some relation to the percentage of ferric oxide in the ash, becoming greatest in ashes with an iron oxide content between 9 and 19 per cent.

Silicate mixtures free from iron oxide did not show any material difference, as may be seen in the following results of tests made with standard pyrometric cones:

TABLE 14.—*Results of tests to determine softening temperatures of standard pyrometric cones*[a] *in H_2–H_2O and CO–CO_2 atmospheres, and in air.*

Cone No.	Per cent of Fe_2O_3 in Seger-cone formula.	Softening temperature in—			Difference.	
		Atmosphere of equal parts of H_2 and H_2O.	Atmosphere of equal parts of CO and CO_2.	Air.	H_2–H_2O and CO–CO_2 atmospheres.	H_2–H_2O atmosphere.
		°C.	*°C.*	*°C.*	*°C.*	*°C.*
06	8.52	1,018	1,026	1,040	+ 8	+22
04	8.54	[b]1,089	1,062	1,085	−27	− 4
02	8.57	1,119	1,119	1,178	+ 0	+59
1	8.58	1,146	1,164	1,182	−18	+36
2	4.38	1,165	1,170	1,181	+ 5	+16
4	None.	1,221	1,240	1,250	+19	+29
6	None.	1,281	1,292	1,286	+11	+ 5
8	None.	[b]1,332	1,330	1,366	− 2	+34
10	None.	[b]1,358	1,340	1,334	−18	−24
12	None.	1,358	1,370	1,367	+12	+ 9

[a] Manufactured by Edward Orton, jr., Columbus, Ohio.
[b] Average of closely checking duplicate determinations.

The average softening-point curves in figure 24 show clearly the effect of increasing percentages of iron oxide in lowering the softening temperatures of coal ashes in reducing atmospheres.

This relation was most clearly indicated in tests with an atmosphere of hydrogen and water vapor, the average softening-point curve dropping from 1,310° to 1,120° C. with an increase of Fe_2O_3 from 5 to 60 per cent. In an oxidizing atmosphere of air the average softening points for the same ashes dropped only from 1,400° to 1,340° C. Hence iron oxide in the ferric form or as magnetite is not to be regarded as having any marked effect in lowering the fusibility of a coal ash. It is only when the conditions are such as to reduce the iron component of a slag to the ferrous form that its effect becomes marked in lowering the temperature of fusion.

All the cones fused in air were strongly magnetic. Those fused in the $CO–CO_2$ atmosphere were mostly nonmagnetic, although a considerable proportion gave evidence of weakly magnetic properties when suspended in the field of a strong electromagnet. In no fused cone was any metallic iron detected under the microscope by the copper sulphate test.

The cones fused in the $H_2–H_2O$ atmospheres were all nonmagnetic except some of those containing more than 40 per cent ferric oxide, which showed traces of metallic iron.

At first thought it would appear that a $CO–CO_2$ atmosphere should unquestionably be accepted as the proper laboratory medium for making fusion tests, as it comprises the principal constituent gases that control slag formation in the fuel bed. There is, however, some difficulty in developing a satisfactory laboratory furnace in which a definite standard concentration of CO and CO_2 can be maintained at temperatures covering the entire range of fusion temperatures of coal ash. For instance, the furnace used in the tests was limited to a maximum temperature of 1,400° C. At higher temperatures the silica tube rapidly crystallized and became permeable to gas. As refractory materials are generally permeable to gases at high temperatures, rapid gas currents would be necessary to maintain any definite atmosphere surrounding the cone.

A further seeming objection to the $CO–CO_2$ atmosphere is the slowness of the reducing and slagging reactions as shown in the experiments made in this atmosphere; approximate equilibrium seemed to be attained more rapidly in the $H_2–H_2O$ atmosphere. It was therefore decided to extend the tests with an atmosphere of hydrogen and water vapor to temperatures above 1,400° C., a modified type of molybdenum furnace being designed for this purpose.

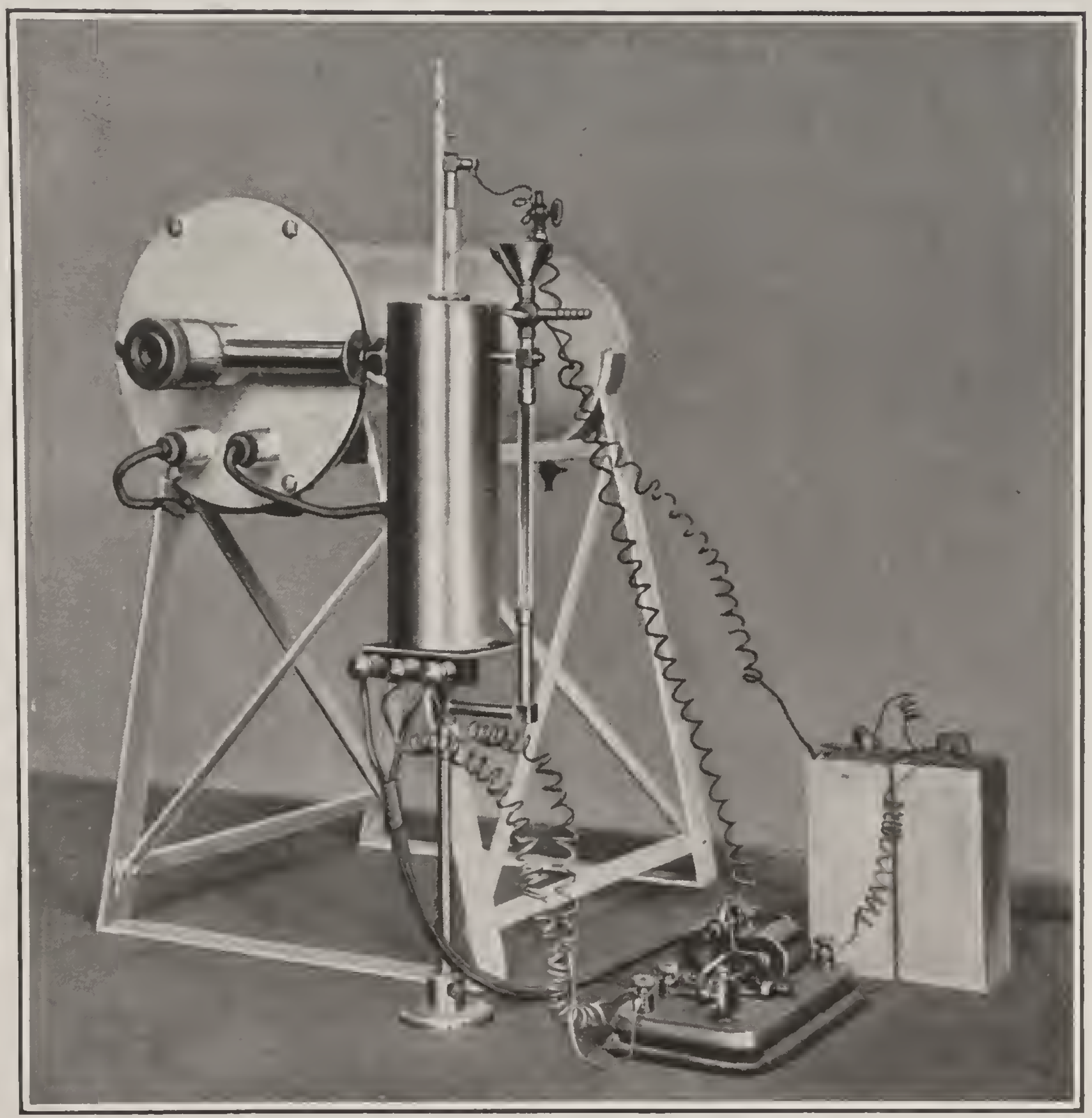
MODIFIED MOLYBDENUM FURNACE AND SATURATOR FOR SOFTENING-POINT DETERMINATIONS IN H_2–H_2O ATMOSPHERE.

In the experiments previously described the ash cone was heated in a glazed vitreosil tube, through which a current of hydrogen and water vapor was passed continuously. The tube was heated in a horizontal platinum-wire resistance furnace. This apparatus, although satisfactory for the more fusible ashes such as were studied in the former experiments, is not adapted to routine work in which samples softening at temperatures above 1,400° C. may be encountered. A molybdenum-wire resistance furnace was therefore designed in which the entire atmosphere inside the gas-tight sheet-iron jacket could be controlled within the required limits of hydrogen and water vapor (30 to 70 per cent water vapor), thus obviating the necessity of a gas-tight refractory tube.

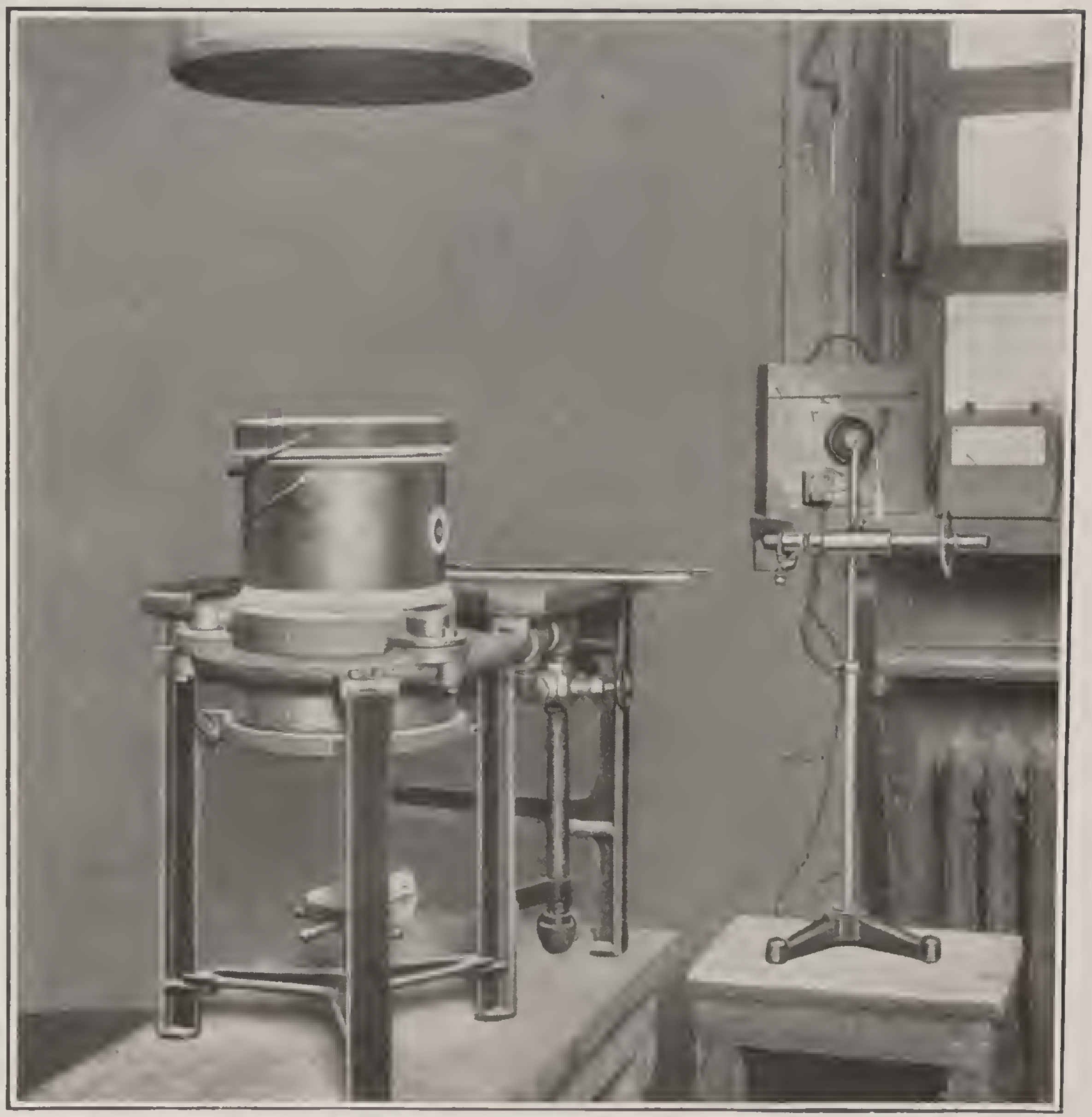
Preliminary tests, in which molybdenum wire was heated to 1,400° C. in various mixtures of hydrogen and water vapor, showed little oxidation of molybdenum in mixtures containing 60 per cent water vapor; at 70 per cent water vapor a slight superficial oxidation was noted, which was little greater at 90 per cent water vapor. Heating even in 100 per cent water vapor to 1,400° C. caused only a thin surface coating of light-colored oxide. On heating in air the metallic molybdenum was completely oxidized at 1,100° C. From these experiments it appeared that no material trouble from oxidation of the molybdenum heating element would be experienced in an atmosphere of equal parts of hydrogen and water vapor within the temperature limits required for softening most coal ashes, at least those containing over 7 per cent ferric oxide, which fuse at a temperature less than 1,500° C. For ashes that soften at a temperature higher than 1,500° C., it is permissible to run pure hydrogen through the furnace, as the iron oxide content is so low as to have little effect on the fusibility.

DESCRIPTION OF FURNACE AND SATURATOR.

The furnace and saturator are shown in Plate IV, *A*. The details are given in figures 26 and 27. The furnace consists in part of an outer shell of No. 14 B. & S. gage sheet iron, 7 inches in diameter and 11 inches long, autogenously welded, gas-tight at the seams, and having at the front a circular 1-inch flange perforated for six equally spaced bolts. Between this flange and the circular end plate, both of which are made of $\frac{1}{4}$-inch sheet iron, is placed an asbestos gasket, $\frac{1}{16}$ inch thick, painted with a thick mixture of powdered graphite and glycerin. This end plate and flange are drawn together sufficiently tight to produce a gas-tight connection. The mixture of hydrogen and water vapor enters the shell through a $\frac{1}{8}$-inch horizontal



A. MOLYBDENUM FURNACE AND SATURATOR.



B. NO. 3 MELTER'S FURNACE ARRANGED FOR FUSING TESTS.

pipe, threaded into a 1¼-inch iron pipe serving as an observation tube, which in turn is threaded into the center of the front end plate. A vertical ⅛-inch iron pipe with a brass stopcock threaded into the top of the observation tube serves as a by-pass around the saturator for admitting pure hydrogen to the furnace. The free end of the observation tube carries a brass sleeve into which is threaded

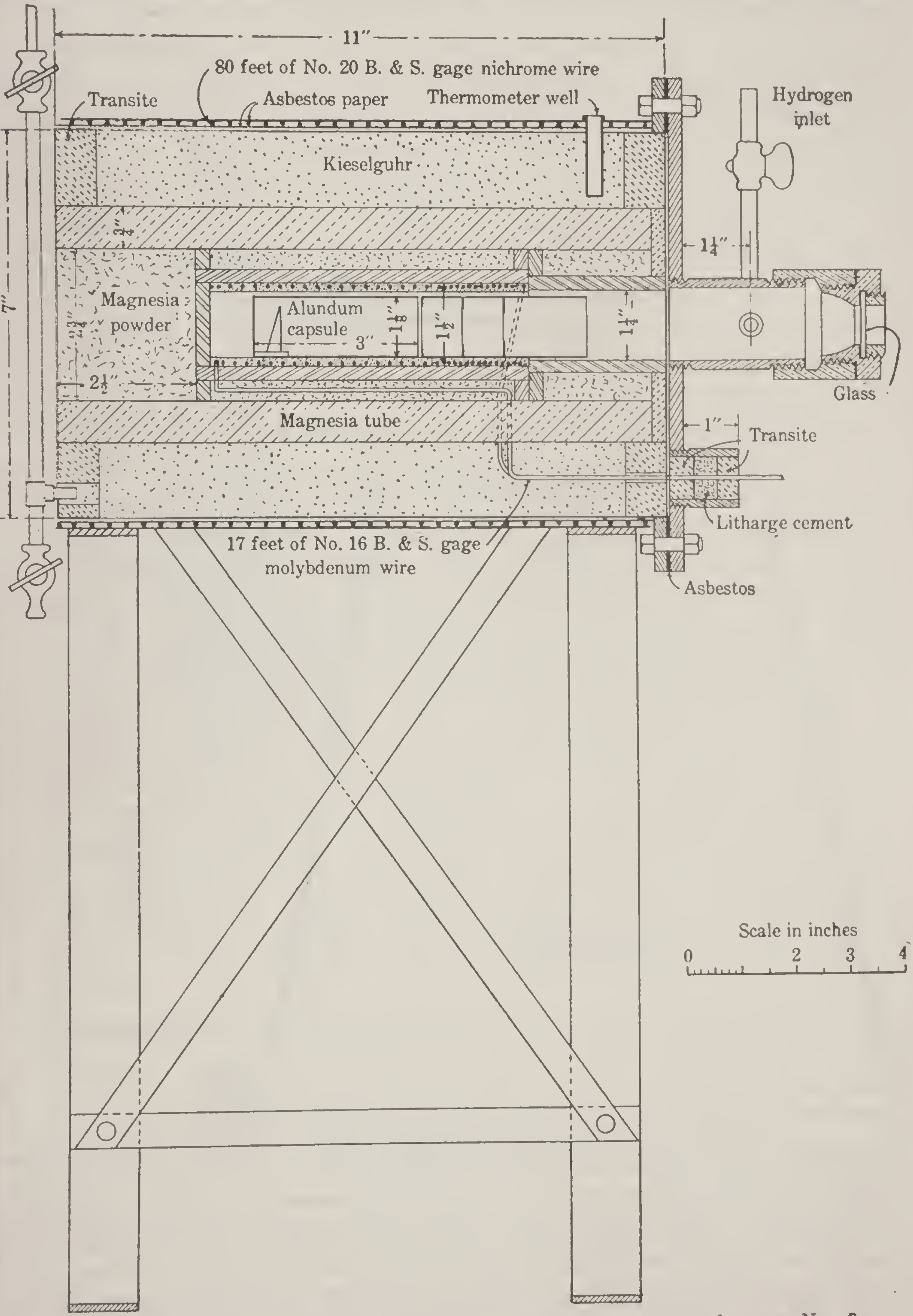


FIGURE 26.—Vertical longitudinal section through molybdenum furnace No. 3.

a brass cap bearing on an asbestos gasket, as shown in figure 26. The cap is fitted with a glass window, through which the temperature is read optically. The glass is held between two asbestos gaskets by

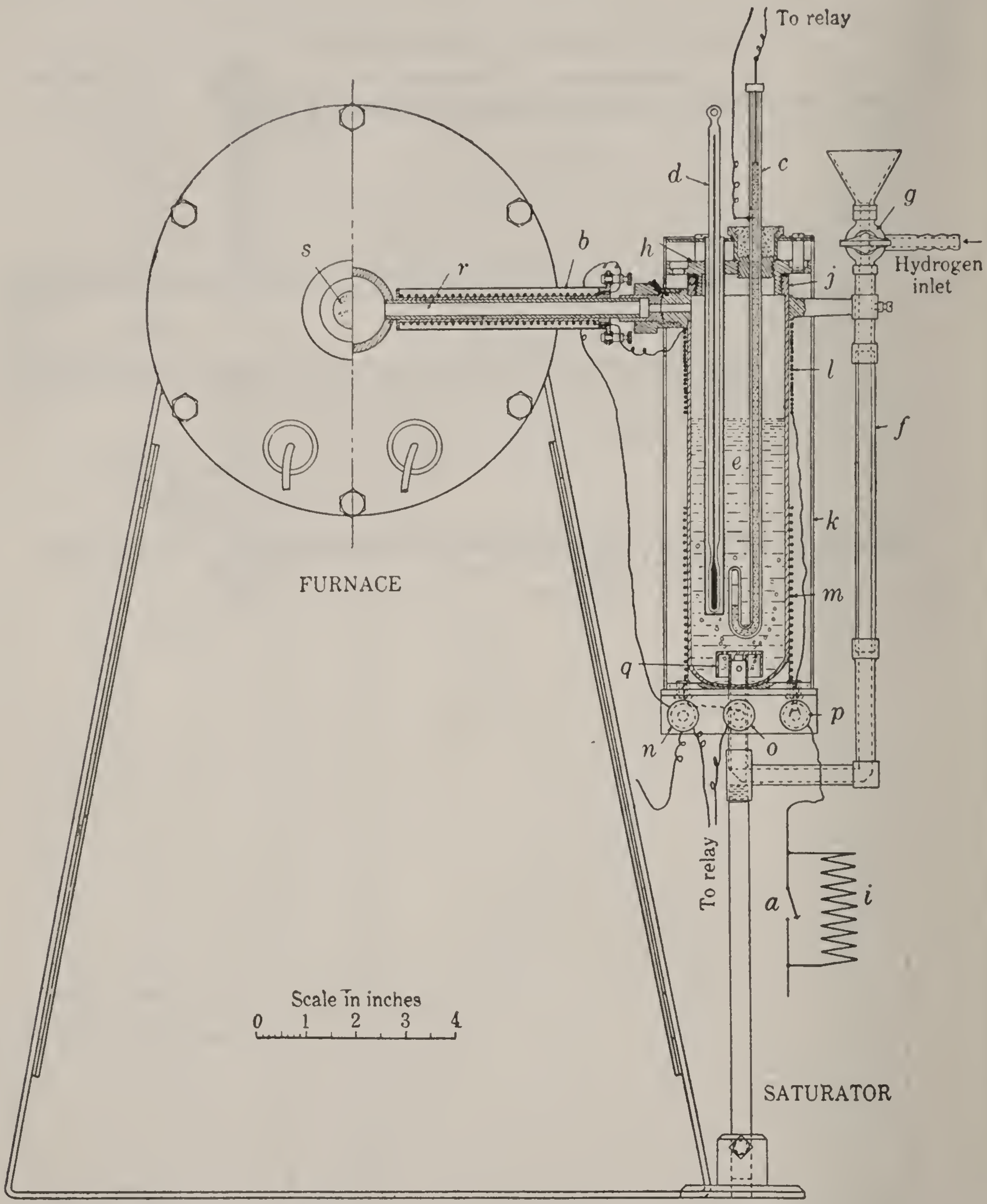


Figure 27.—Front elevation of molybdenum furnace No. 3 and vertical section through saturator. *a*, Switch for cutting out resistance *i*; *b*, metal jacket covering superheater; *c*, vapor-pressure thermoregulator; *d*, thermometer; *e*, water in saturator; *f*, glass gage; *g*, brass three-way cock; *h*, cover of saturator; *i*, resistance coil; *j*, body or saturator; *l*, upper heating coil of saturator for continuous duty, *m*, lower heating coil of saturator for intermittent duty; *n*, *o*, *p*, binding posts; *q*, perforated baffle for breaking up hydrogen into small bubbles; *r*, ⅛-inch iron pipe wound with asbestos and nichrome wire to prevent condensation of water vapor; *s*, glass window for observing cones.

a threaded brass ring. To insert a cone in the furnace, the entire cap is unscrewed from the brass sleeve. The furnace is supported in an iron frame so that its center is about 17 inches above the table, which is a convenient height for optical measurements of temperature.

The heating element consists of 17 feet of molybdenum wire, No. 16 B. & S. gage, wound upon the inside of an alundum tube, 1½ inches in internal diameter, 6 inches long, and closed at one end by an alundum disk cemented in with alundum cement. The wire is wound 6 turns to the inch except at the front, where compensation for radiation is obtained by winding 10 turns on the first inch. This tube, carrying the coil of molybdenum wire, when placed in position, lies in the axis of the cylindrical shell, 2½ inches from the rear end and 2½ inches from the front end, measured to the plane of the flange. The heated alundum tube is supported by being placed within a magnetic tube 2¾ inches in internal diameter and 10¾ inches long. The magnesite tube prevented deformation of the alundum tube at high temperatures. The narrow annular space between the tube carrying the molybdenum coil and the magnesite tube is filled with light calcined magnesia powder, free from carbonate.

The magnesite tube is provided with two slots at the front end for leading out the terminals of the molybdenum coil. At the front end the cylindrical heating chamber is continued along the axis by a short alundum tube, 1¼ inches in internal diameter. The 1¼-inch annular space between the magnesite tube and the outer iron shell is filled with previously ignited kieselguhr (infusorial earth). The magnesite tube is supported at the ends by rings of "transite," 1 inch thick. The rear terminal of the heating coil passes between the alundum tube and the magnesite tube to the front end of the heating coil, through a slot in the magnesite tube, and out of the furnace through a ¾-inch pipe, 1 inch long, threaded into the front plate. The front terminal passes through a similar slot in the magnesite tube and out of the furnace. The positions of these terminals can be better seen by reference to figure 27, which shows the front of the furnace.

The leads are centered in the pipes by an outer and an inner disk of "transite," between which is a layer of litharge cement. (See fig. 26.) This makes a gas-tight arrangement, which is not affected by the temperature to which the shell is subjected.

In order to obtain a background suitable for measuring the temperature of the test piece with accuracy, a thin alundum disk is cemented in at the rear end of the tube carrying the heating coil. The space back of this disk is provided with light calcined magnesia powder. The heating space, which is 1¼ inches in diameter and 6 inches long, affords a uniformly heated part approximately 3 inches long. The ash cone is placed in a No. 5811 alundum extraction thimble, 1⅛ inches by 3 inches. When this is to be inserted the brass cap on the observation tube is taken off and the thimble is pushed back about 1 inch beyond the center of the heating coil. Another alundum capsule having two or three diaphragms is then introduced to minimize radiation loss. Under normal conditions the cone is barely visible against the background.

In order to heat the outer shell of the furnace rapidly to a temperature at which no water can condense from the atmosphere of hydrogen and water vapor in the furnace, 110 feet of No. 20 B. & S. gage nichrome wire, about five turns per inch, is wrapped around the outside of the furnace. This length and size of wire will allow three amperes to flow when the wire is connected directly across a 220-volt circuit and will heat the outer shell to a temperature of 90° C. in 15 minutes, which will prevent the condensation of moisture from the H_2–H_2O atmosphere.

The wire is insulated from the furnace by first wrapping the iron shell with thin asbestos paper. The resistance wire is wound on this paper and fastened at each end of the furnace with asbestos cord. Another sheet of asbestos paper is then wrapped around the furnace to cover the resistance wire, as shown in figure 26.

The desired mixture of hydrogen and water vapor is obtained by bubbling hydrogen through water held at the proper temperature. The exact temperature depends on the prevailing barometric pressure and the proportions of water vapor and hydrogen desired. A temperature of 81° C. is required for a mixture containing equal parts of water vapor and hydrogen.

The saturator is constructed of brass and its temperature is controlled automatically by a vapor-pressure thermo regulator. The water in the saturator is heated by a coil of nichrome wire wound around the saturator and connected in series with a relay which is operated by the thermo regulator. Complete details of this saturator and an earlier form of the molybdenum furnace have been published in a previous report.[a] The method of operating the present form of molybdenum furnace in making a softening-temperature determination is essentially the same as described therein.

Similar results were obtained on repeating in this furnace softening-point determinations of a number of ashes that had been tested previously in the silica-tube furnace, shown in figure 17 (p. 51).

This furnace when operated as described above gave softening temperatures that were usually duplicated within the known errors of observation (about 25° C.). The fused cones from coal ash containing less than 50 per cent ferric oxide did not show any material degree of magnetism in the field of a strong electromagnet, indicating a reduction of the iron present in the ash to the ferrous state, but not to metallic iron. This condition corresponds to the minimum softening temperature, for the determination of which the furnace was designed. Most of the determinations in hydrogen and water vapor given in Table 13 were determined in this furnace, and it was given a thorough trial in routine work and in continuing the study of factors affecting the minimum softening temperature in atmospheres of hydrogen and water vapor.

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[a] Fieldner, A. C., and Feild, A. L., A new method and furnace for the determination of the softening temperature of coal ash under fuel-bed conditions: Jour. Ind. and Eng. Chem., vol. 7, 1915, p. 829.

EXPERIMENTS WITH VERTICAL AND HORIZONTAL PLACING OF CONES IN DETERMINATION OF SOFTENING TEMPERATURE.

Ricketts,[a] of the New York Edison Co., has modified the usual Seger-cone method of ash-fusing determinations by placing the cone in a horizontal position. Marks also prefers this method as being more sensitive and as giving indications which can be duplicated more accurately than the usual vertical method.[b] His objection to the vertical position was based on the seeming separation of a more fusible and very fluid constituent in certain ashes, which ran downward to the base of the cone and left the apex seemingly unchanged in a vertical or slightly inclined position until at a higher temperature the apex also melted.

On the other hand, Palmenberg[c] states that "this difficulty does not seem to present itself when the cones are made long and thin, at least not to the extent of interfering with the tests. If the ash is thoroughly ground and mixed, a complete fusion of the whole mass should take place."

A large number of determinations made with vertical cones in the Bureau of Mines laboratory did not show any separation of a fluid constituent when the determination was made in a mixture of equal parts of hydrogen and water vapor or a similar mixture of CO and CO_2, nor has there been any such separation of a low-melting slag in any coal ash containing less than 50 per cent ferric oxide when the determination has been made in the purely oxidizing atmosphere of a platinum-wire resistance furnace. However, in some coal ashes, as fused in several types of carbon and graphite resistance furnaces containing a strongly reducing atmosphere of carbon monoxide and nitrogen, certain phenomena observed were in general similar to the observations of Marks. These cones, as shown in Plate I, *C* (p. 42), did not deform in the usual manner. The upper part of the skeleton of a fused cone contained a considerable quantity of metallic iron and consisted simply of the unchanged original apex of the cone. The thickened lower part was a thin-shelled bubble of fused slag. Hence the authors consider that this phenomenon was not due to a separation of fusible constituents, but merely to an earlier fusion of the thick part of the cone. The thin part was rendered less fusible more rapidly than the thick part by reduction of iron oxide to the metallic iron.

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[a] Ricketts, E. B., Discussion of L. S. Marks's paper on "The clinkering of coal": Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 213; Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 830–833.

[b] Marks, L. S., The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 806–807; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 206.

[c] Palmenberg, O. W., Discussion of paper on "Clinkering of coal," by L. S. Marks: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 825; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, pp. 211–212.

Although the objection cited by Marks to the use of the vertical position does not apply to softening-temperature determinations made in an oxidizing atmosphere nor in a mixture of equal parts of hydrogen and water vapor or of carbon monoxide and carbon dioxide, a number of fusibility determinations with cones placed horizontally have been made to determine whether horizontal placing in itself possessed any features that would make such placing preferable in routine testing to the usual vertical placing. The effect of inclination of the ash cone on the softening temperature has been shown previously for three different degrees of inclination, viz, 25°, 35°, and 45° from the vertical. Although the effect of the 25° inclination was negligible, the 35° and 45° inclinations gave average softening points about 20° lower than those obtained with cones in the vertical position. Undoubtedly the horizontally mounted cone should give still lower values. Marks[a] made a series of tests with cones placed horizontally and vertically in a Meker furnace having an oxidizing atmosphere. He found that the deformation temperature of a horizontal cone was always lower than of a vertical cone. The difference varied from 5° to 50° C. and was less for the more fluid melts. In making the tests with the cones in the horizontal position he took as the "down point" the temperature at which the tip of the cone pointed vertically downward.

METHOD OF CONDUCTING EXPERIMENTS.

Tests with cones in both the horizontal and the vertical position were made in the new type molybdenum furnace with an atmosphere containing equal parts of hydrogen and water vapor. The cones were three-sixteenths of an inch wide at the base by 1 inch high, and were heated at the usual rate of 5° and 2° C. per minute.

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[a] Marks, L. S., The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 810; Jour. Am. Soc. Mech. Eng., vol. 37, 1915, p. 206.

In tests with the cone in the horizontal position the cone was placed at right angles to the axis of the furnace, a position favoring easy observation of the softening process. The cone was supported by placing its base in a notch cut in the wall of an alundum capsule, which fitted snugly within the larger capsule used in the tests in which the cone was placed vertically. The small inner capsule was sliced longitudinally slightly above its axis, and the upper "half" was removed. Thus, the cone could be placed in position in the notch and secured by a fillet of the usual infusible mixture of kaolin and alumina. After the cone had dried, the capsule could be placed within the larger capsule without disturbing the mounted cone.

It was found just as difficult to define the down points of horizontal cones as to fix those of vertical cones. Some of the ash cones formed globules at the tips and shortened considerably in length before the axes bent appreciably; the tips of others bent vertically for only a short distance toward the bases; in a third type, the tip of the cone did not become vertical until the larger part of the cone axis was vertical or nearly vertical. In this work, therefore, the "down point" or softening point of cones placed horizontally is defined as the temperature at which a vertical line may be drawn through the tip of the cone or the center of the globule formed at the top, and a point on the axis of the cone midway between the tip and the base. Likewise a 45° deformation for horizontal cones is reached when a similar line makes an angle of 45° with the vertical.

The "softening point" or "down point" for vertical cones is the temperature at which the tip of the bent cone touches the base, or, if it fails to bend, the temperature at which the cone settles into a more or less spherical lump, which may or may not have swelled.

RESULTS OF EXPERIMENTS.

The results of softening-temperature determinations of seven synthetic mixtures and eight coal ashes with cones placed both vertically and horizontally are given in Table 15. The synthetic mixtures were made from pure North Carolina kaolin, ferric oxide, and calcium carbonate. The analyses of the mixtures and the ash samples are given in Table 16.

TABLE 15.—*Results of softening-temperature tests, showing relative effects of vertical and horizontal placing of cones.*

[Values in centigrade degrees.]

Sample No.	Cones placed vertically.			Cones placed horizontally.			Difference.		
	Temperature of initial deformation.	Down point	Softening interval.	Temperature of initial deformation.	Down point.	Softening interval.	Temperature of initial deformation.	Down point.	Softening interval.
M10	1,183	1,209	26	1,135	1,170	35	48	39	− 9
M23	1,103	1,105	2	1,072	1,113	41	31	−8	− 39
M33	1,063	1,086	23	1,053	1,063	10	10	23	13
M20	1,204	1,403	199	1,191	1,204	15	13	199	184
M19	1,401	1,408	7	1,216	[a]1,394	178	185	14	−171
M4	1,431	1,431	0	1,421	[a]1,428	7	10	3	− 7
M28	1,340	1,426	86	1,242	1,340	98	98	86	− 12
20913	1,164	1,202	38	1,113 1,089	1,177 1,191	64 102	63	18	− 45
19472	1,153	1,204	51	1,113 1,113	1,158 1,165	45 52	40	42	2
20914	1,080	1,116	36	1,051 1,058	1,113 1,096	62 38	25	11	− 14
16584	1,291	1,351	60	1,177	1,212	35	114	139	25
20915	1,057	1,099	42	1,044	1,063	19	13	36	23
15842	1,350	1,409	59	1,145	1,291	146	205	118	− 87
19160	1,504	1,551	47	1,482	[a]1,551	69	22	0	− 22
20233	1,320	1,378	58	1,127	[a]1,362	235	193	16	− 17
Average	....	....	49	....	....	70	71	49	− 22

[a] Cone did not bend vertically downward, but simply shortened up to a globule in the same manner as did cones mounted vertically.

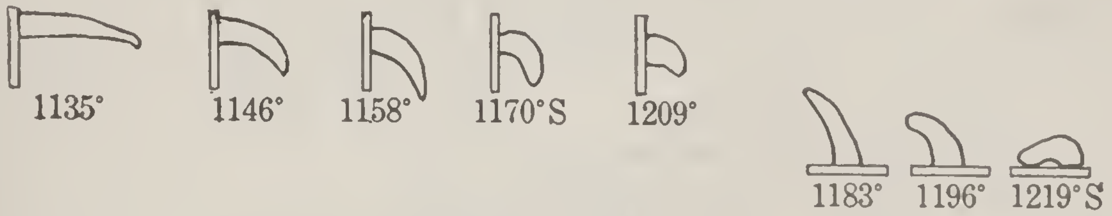
Temperatures at Which Swelling of Cones Became Noticeable.

Sample No.	Cone placed vertically.	Cone placed horizontally.
	°C	°C
M33	1,063	1,058
20913	1,164	1,130
19472	1,153	1,165
16584	1,291	1,191
20915	1,057	1,055

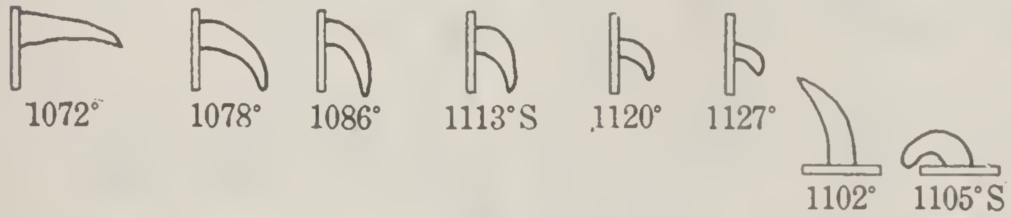
TABLE 16.—*Results of analyses of coal ashes and synthetic mixtures tested with the cones used in the vertical and in the horizontal position.*

Sample No.	SiO_2.	Al_2O_3.	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.	Total.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
M10	32.6	27.9	32.7	....	6.9	0.04	....	....	....	100.1
M23	32.0	27.4	40.7	....	.03	.04	....	....	....	100.2
M33	38.1	21.4	35.5	1.3	.7	.7	0.5	2.1	0.3	100.6
M19	38.3	32.9	28.8	....	.04	.05	....	....	....	100.1
M4	45.6	39.1	5.9	....	9.6	.1	....	....	....	100.3
M28	53.3	29.9	9.7	1.8	.9	.9	.6	2.9	.4	100.4
20913	43.3	31.4	13.6	1.2	4.2	1.4	.6	2.9	1.4	100.0
19472	38.4	24.2	22.4	1.1	7.7	.9	.3	1.9	3.8	100.7
20914	35.7	23.5	32.9	1.2	3.2	1.1	.3	1.1	.5	99.5
16584	54.8	29.2	6.9	1.8	1.4	.6	1.9	2.1	1.0	99.7
20915	12.3	12.2	69.7	.4	3.9	.7	.3	.6	.2	100.3
15842	55.8	33.5	5.0	.9	1.5	.7	.5	2.2	.1	100.2
19160	54.2	33.0	7.4	....	1.1	.5	1.0	1.9	.9	100.0
20233	47.3	34.6	9.8	1.8	1.3	.4	2.1	2.5	.1	99.9

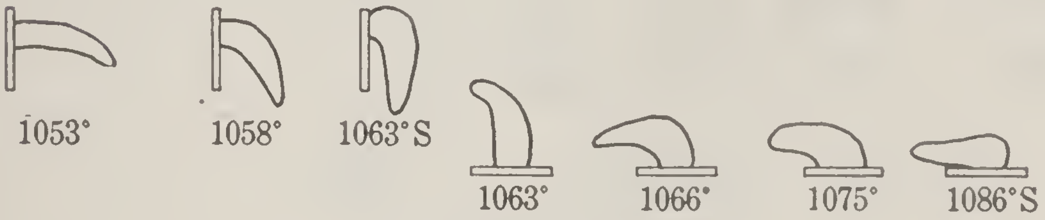
As would be expected, the cones, placed horizontally, had lower softening points than those placed vertically, the differences, varying with the viscosities of the melting eutectics, being 0° to nearly 200° C. The average difference for the 15 samples tested was 49° C. Owing to the greater force acting on the cross section of a cone placed horizontally, it begins bending at a comparatively early stage of the melting process, so that the softening interval is usually somewhat longer than that of a cone placed vertically.



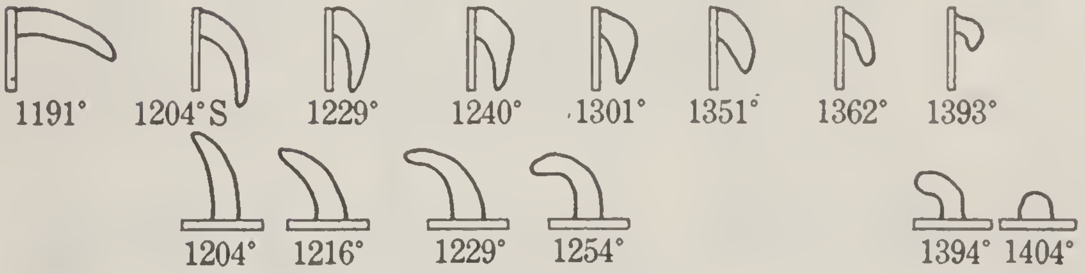
Sample No. M10



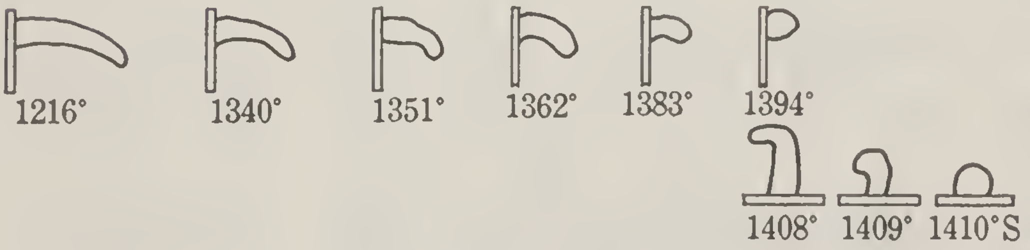
Sample No. M23



Sample No, M33



Sample No. M20



Sample No. M19

FIGURE 28.—Appearance of vertical and horizontal ash cones at various stages of softening—samples M10, M19, M20, M23, and M33.

Figures 28, 29, and 30 show the appearance of the horizontal and the vertical cones at various stages of softening.

Some of these samples, as M23, M33, M4, 20914, 19160, and 20915, showed for either position final deformation points that were not far apart on the temperature scale, whereas other samples, as M20, 15842, and 16584, reached the so-called "down point" for the horizontal position at a temperature of 100° to 200° C. below the "down point" for the vertical position. In particular, the horizontally placed cone

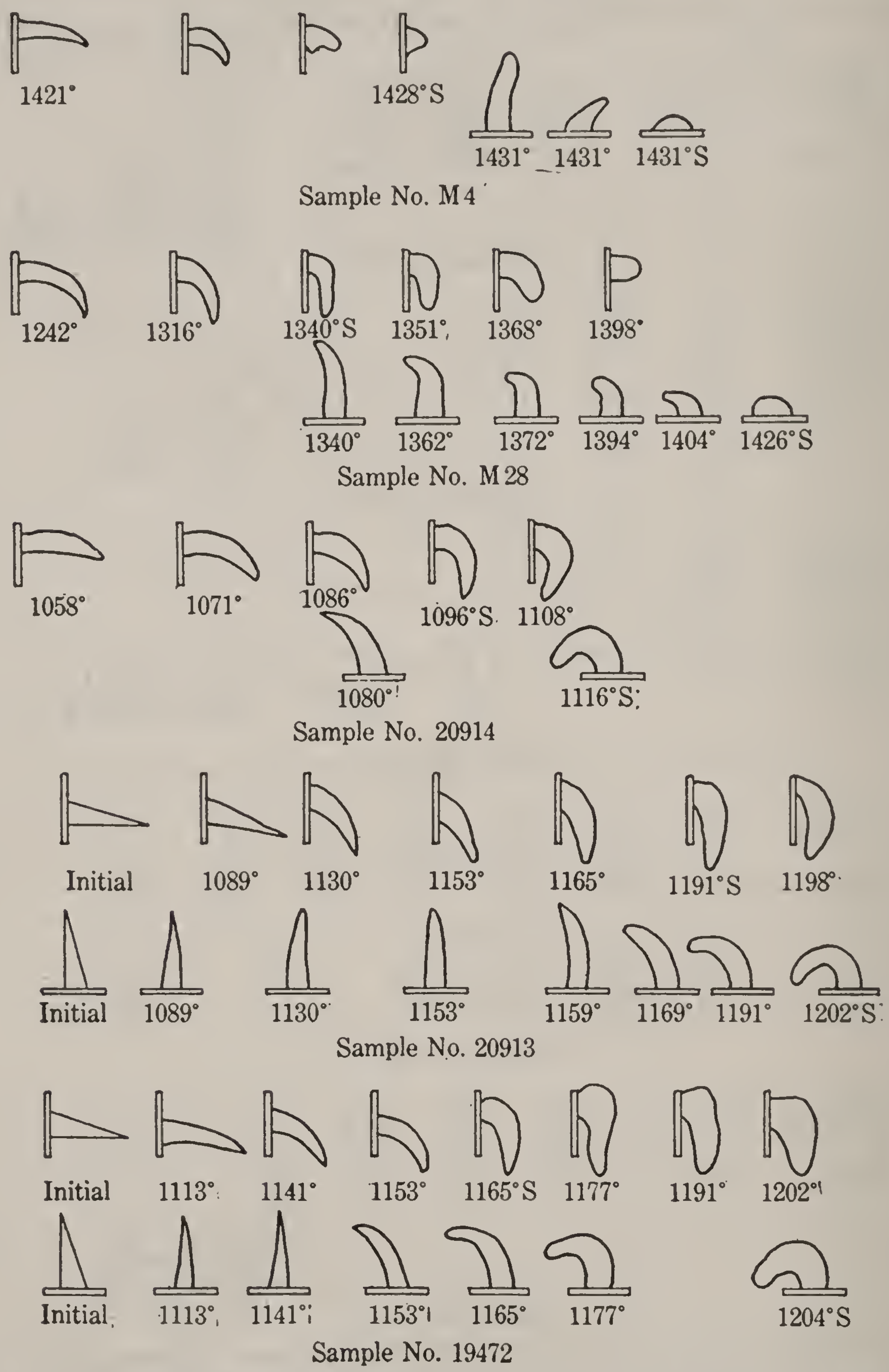
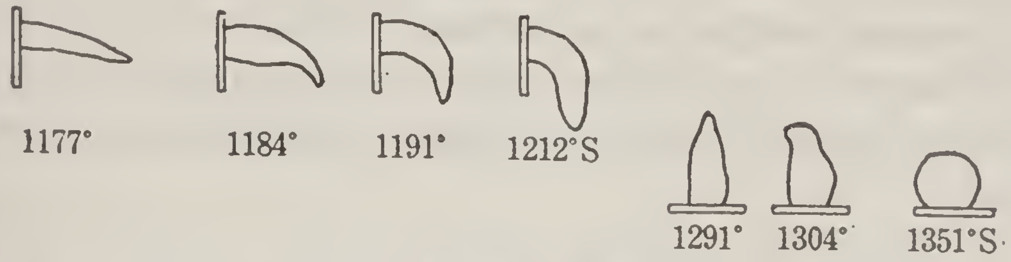


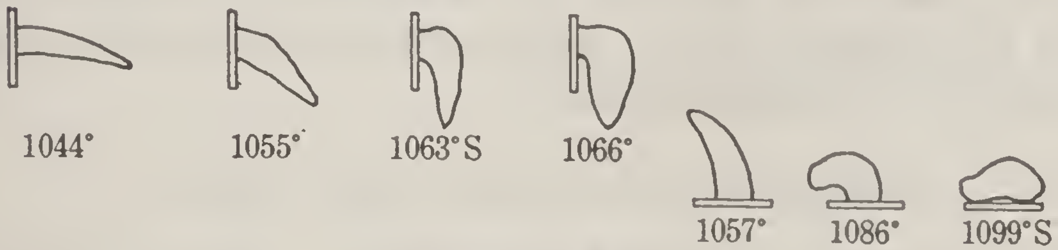
FIGURE 29.—Appearance of vertical and horizontal ash cones at various stages of softening—samples M4, M28, 19472, 20913, and 20914.

from sample 16584 reached the "down point" before any visible deformation took place in the cone placed vertically.

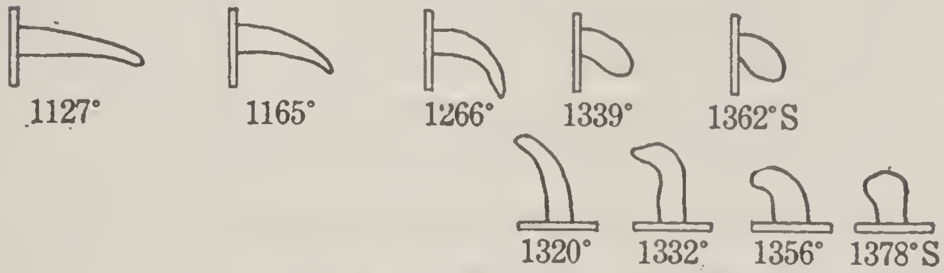
This difference in behavior illustrates the extremely variable character of the fusion process in coal ashes. The sudden formation of a relatively large proportion of fluid eutectic would cause the cone to deform rapidly in either position and at similar temperatures. Sample M4 is an especially good example of such a mixture. The softening interval in either position was not over 7° C., and the down point for either position was practically the same.



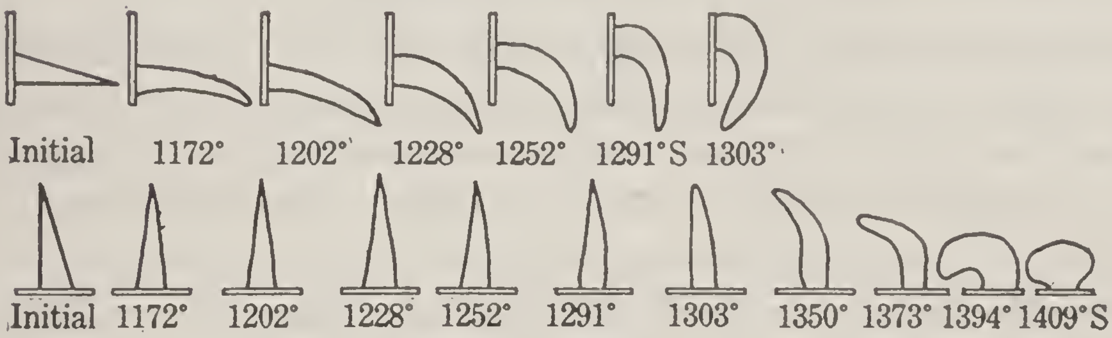
Sample No. 16584



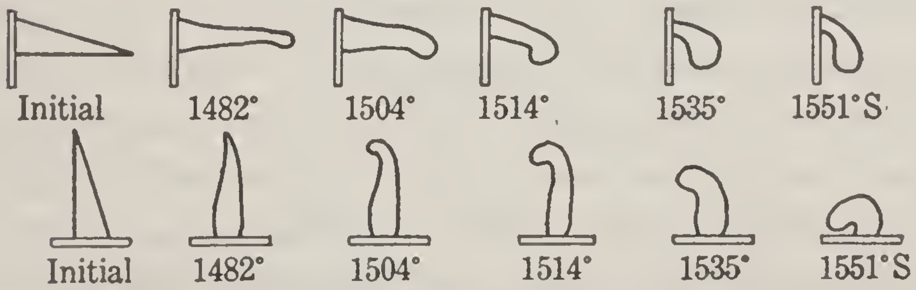
Sample No. 20915



Sample No. 20233



Sample No. 15842



Sample No. 19160

FIGURE 30.—Appearance of vertical and horizontal ash cones at various stages of softening—samples 15842, 16584, 19160, 20233, and 20915.

Sample M19 represents another type of fusion characterized by certain peculiar relations between surface tension and viscosity. The cone, whether in the horizontal or the vertical position, fused to a nearly spherical globule without bending in the usual manner.

It is evident that no fixed difference can be assigned between results obtained by vertical mounting of the cones as compared with horizontal mounting, nor is there any apparent advantage as regards closeness of duplication inherent in either method. The principal difference is that the down point of a vertical cone indicates a temperature at which the melting ash has reached a greater degree of fluidity than that indicated by the deformation point of a horizontal cone. This is undoubtedly a point in favor of the vertically mounted cone, as, according to furnace data thus far accumulated, clinker does not cause serious trouble unless it becomes sufficiently fluid to flow in the fuel bed.

EFFECT OF DEXTRIN BINDER OR RESIDUAL CARBON ON SOFTENING TEMPERATURE OF ASH, AND ITS RELATION TO INTUMESCENCE OF SOFTENING CONE.

SWELLING OF MELTING ASH CONES.

Attention has been called to the pronounced vesicular structure of ash cones fused in a hydrogen atmosphere. A similar decided "bleb" structure, accompanied in many instances by considerable intumescence, has been noted in the majority of ash samples that were fused in mixtures of hydrogen and water vapor. Indeed, some degree of swelling of the softening cone is characteristic of fusions made in any atmosphere containing material proportions of hydrogen. Swelling and intumescence, but to a distinctly less degree, occur also in atmospheres containing carbon monoxide. In atmospheres of air, water vapor, or carbon monoxide intumescence occurs rarely. It appears, then, that intumescence occurs most generally under conditions favoring the reduction of the original ferric oxide of the coal ash.

As other possible causes of swelling there may be considered: (1) Presence of substances that develop an appreciable vapor pressure at the melting temperature; (2) decomposition of sulphates, sulphides, and carbonates; and (3) oxidation of unburned particles of coke in the interior of the cone. Cause 3 especially should receive careful consideration, as even after prolonged heating in a muffle at 750° C. coal ash sometimes contains minute particles of coke, which are visible under the microscope. Unburned dextrin or other organic binding materials may also under certain conditions cause the melting cone to swell.

EXPERIMENTS TO DETERMINE EFFECT OF UNBURNED CARBON IN ASH ON SOFTENING OF CONE.

In order to determine the effect of carbonaceous binder and unburned carbon on the fusion of cones, several samples of various types of ash were reground to a fine powder and reheated five or six hours in an electric muffle at approximately 800° C. When, after this treatment, the ash still showed possible traces of coke, it was again ground in the mortar and heated for several hours in a Meker muffle furnace at a temperature of 900° to 1,000° C., until microscopic examination showed complete absence of all traces of coke. Some of these ashes were slightly "fritted" at this temperature so that repulverization was necessary before molding the cones. The results of analyses of these samples are presented in Table 17 following:

TABLE 17.—*Results of analyses of coal ashes used in experiments to determine influence of carbon on the softening temperature and the intumescence of ash cones.*

Sample No.	SiO_2.	Al_2O_3.[a]	Fe_2O_3.	TiO_2.	CaO.	MgO.	Na_2O.	K_2O.	SO_3.
	P.ct.	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*
86	31.0	22.7	42.8	....	2.1	0.5	....	....	0.6
101	42.1	32.7	19.3	....	2.5	.6	0.9	1.3	1.1
15	54.1	24.8	9.4	2.3	4.0	1.4	1.0	.8	2.8
9	54.8	27.0	7.0	1.3	4.3	1.7	.3	3.1	.6
102	56.9	27.8	9.3	....	1.6	1.1	1.1	1.3	.9
103	52.2	33.4	7.9	....	1.9	.6	1.5	1.4	.9
7	54.1	34.7	4.5	1.5	1.2	.9	.6	2.5	.2

[a] Includes P_2O_5 and TiO_2 where not separately determined.

Seven samples prepared in this way were made into three series of cones, as follows: The cones in series 1 were molded with distilled water only, so that no carbonaceous matter could be present; the cones in series 2 were molded in the usual manner with a 10 per cent dextrin solution; the cones were placed in the furnace without previous burning of the dextrin; the cones in series 3 were molded with 10 per cent dextrin solution, after an amount of sugar carbon equal to 10 per cent of the mixture had been added to the ash; these cones were also placed directly in the furnace without any previous ignition.

One series of softening-temperature tests was made in an atmosphere of equal parts of hydrogen and water vapor, in the furnace shown in Plate IV, *A* (p. 74); another series of tests covering ashes that softened under 1,400° C. was made in an atmosphere containing equal parts of CO and CO_2, in the furnace shown in Plate II (p. 46). The softening temperatures and the degree of intumescence observed are given in Table 18 following.

TABLE 18.—*Results of tests to determine influence of carbon on the softening temperature and on the intumescence of ash cones.*

Rate of temperature increase 5° C. per minute until initial softening of cone; then 2° C. per minute until cone melted.]

ATMOSPHERE OF EQUAL PARTS OF HYDROGEN AND WATER VAPOR.

Lab. No.	Softening temperature.			Differences.		Degree of intumescence.		
	Series 1—cone molded with distilled water only	Series 2—cone molded with 10 per cent dextrin solution.	Series 3—ash containing 10 per cent sugar carbon and dextrin.	Between results of series 1 and 2.	Between results of series 1 and 3.	Series 1—distilled water only.	Series 2—10 per cent dextrin solution.	Series 3—10 per cent sugar carbon and dextrin.
	° *C.*	° *C.*	° *C.*	° *C.*	° *C.*			
22665	1,080	1,072	1,060	− 8	−20	Marked	Marked	Marked.
19159	1,146	1,182	1,177	+36	+31	do.	do.	Do.
16585	1,278	1,291	1,254	+13	−24	Slight	Slight	Slight.
15848	1,325	1,328	1,326	+ 3	+ 1	Moderate	Moderate	Moderate.
19155	1,345	1,351	1,384	+ 6	+39	Slight	Slight	Do.
19161	1,500	1,500	1,512	0	+12	do.	do.	Slight.
15846	1,600	1,605	1,596	+ 5	− 4	do.	do.	Do.
Average.				+ 8	+. 5			
ATMOSPHERE OF EQUAL PARTS OF CO AND CO_2.								
22665	1,077	1,077	1,085	0	+ 8	None	None	Very slight.
19159	1,350	1,350	1,320	0	−30	do.	do.	Do.
16585	1,301	1,292	1,279	− 9	−22	do.	do.	None.
15848	1,340	1,345	1,320	+ 5	−20	Slight	Moderate	Slight.
Average.				− 1	−16			

It may be concluded from the results of the tests that the puffing or intumescence is not due primarily to the presence of unburned coke or carbon particles in the coal ash. At the rate of heat increase used in the experiments, namely, 5° C. per minute, small amounts of carbon are probably removed from the system by reacting with the gaseous atmosphere before the sintering temperature is reached. In an atmosphere containing oxygen, 50 per cent water vapor, or 50 per cent carbon dioxide, the solid carbon will be removed as carbon monoxide at temperatures below 1,000° C., provided the rate of heating is not too rapid. This conclusion is also corroborated by the experiments of Marks and those of Palmenberg. Marks[a] has shown that ash cones molded with 10 per cent dextrin solution softened in the oxidizing atmosphere of a Meker furnace at practically the same temperature as similar cones molded with distilled water only. The rate of heat increase was 2° C. per minute.

Palmenberg,[b] who also used a Meker furnace, found that as much as 10 per cent carbon had no appreciable effect on the softening temperature. A few investigators have reported that traces of carbon left in the ash or the presence of an unoxidized organic binder caused intumescence and abnormal deformation points. However, an analysis of their methods of making softening-point determinations has shown that they invariably heated the cone rapidly—40° to 70° C. per minute. Under these conditions it is quite possible that carbon was not removed by the surrounding gases before the softening temperature was reached.

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[a] Marks, L. S., The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 810.

[b] Palmenberg, O. W., Discussion of Marks's paper, The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, p. 824.

MOLYBDENUM FURNACE AS STANDARD FURNACE FOR FUSION TESTS.

The molybdenum furnace was used for a period of one year in routine and in special softening-temperature determinations of coal ash, clays, and similar materials, thus affording an excellent opportunity to pass judgment on its practical usefulness as a standard type of furnace for determining the fusibility of coal ash. Any standard method for the determination of the fusibility of coal ash should answer the following requirements:

(1) The atmosphere should reduce the iron component of the ash to the ferrous state in which it can exert its maximum fluxing action.

(2) The method should give consistent and reproducable results when used by different chemists.

(3) The method should be simple and rapid.

(4) The furnace should be comparatively cheap and easily operated and repaired with the means available in the ordinary commercial laboratory.

If these criteria be applied to the molybdenum furnace with an atmosphere of hydrogen and water vapor as tried for a year in the Bureau of Mines laboratory, the method may be said to meet requirements 1 and 2, but to fail with respect to requirements 3 and 4.

In recent years the price of molybdenum wire has advanced materially so that the cost of renewing heating elements is a serious item. With temperatures up to 1,450° C. the element is fairly durable, but if the furnace is frequently heated to higher temperatures, as 1,600° C., oxidation by the water vapor causes the molybdenum wire to waste away and eventually burn out.

The replacement of such units, although comparatively unimportant in a laboratory equipped for renewing electric furnaces, is a more serious matter to the busy commercial laboratory. Such laboratories do not have the time nor the means to make the various electrical repairs and adjustments required in the somewhat complicated electrical system of the molybdenum furnace and satuator.

A simple form of gas furnace would be cheaper and more easily repaired; it could also be adapted to making a number of determinations simultaneously. Hence the possibilities of the gas-heated furnace were studied to determine whether the combustion could be so controlled as to produce an atmosphere of approximately constant composition and of the proper reducing character.

FUSIBILITY OF COAL ASH AS DETERMINED IN GAS FURNACE.

THEORETICAL CONSIDERATIONS.

As mentioned previously, exceedingly variable results for softening temperature have been obtained in tests made in muffle furnaces. These variations were due to the variable amounts of gas and products of combustion that penetrated the muffle and reacted with the coal ash. Now, if a furnace be constructed with a very porous muffle or one with suitable openings so that the combustion gases have free access to the cones, it should be possible to subject the cones to a reducing atmosphere merely by operating the furnace with a deficient air supply. The proportion of reducing gases in the furnace atmosphere could thus be increased to a certain maximum limit at which combustion slows down to a rate at which the desired temperature can not be attained. If this maximum amount of reducing gas does not reduce the iron constituents of the ash to metallic iron, the most desirable method of operating the gas furnace becomes simple, namely, using the lowest possible proportion of air in the mixture that will sustain the required intensity of combustion.

In the experiments described subsequently no more than traces of metallic iron were ever found under maximum reducing conditions.

DESCRIPTION OF FURNACE.

An investigation of the various gas furnaces on the market led to the selection of the No. 3 melter's furnace (Pl. IV, *B*, p. 74) of the American Gas Furnace Co. Ricketts[a] has used this furnace for a number of years for determining ash-softening temperatures in the laboratory of the New York Edison Co. This type of pot furnace is especially suitable for fusion determinations in that the burners, three in number, are on a tangent near the base of the furnace, thus producing a rotary flame which completely surrounds the crucible in which the cones are placed. The whirling flame heats the crucible uniformly and when the furnace is operated with excess gas supply, a reducing atmosphere is maintained within the crucible.

The furnace proper consists of three easily replaceable fire-clay parts, namely, a lower cylinder containing the three tangential tuyères, and forming the bottom of the furnace, a removable upper cylinder 7 inches in internal diameter and 7 inches high, and a cover plate 1$\frac{1}{2}$ inches thick, having a vent hole in the center for the flue gas.

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[a] Ricketts, E. B., Discussion of paper by L. Marks, The clinkering of coal: Trans. Am. Soc. Mech. Eng., vol. 36, 1914, pp. 801–833.

The manufacturers were asked to modify the stock design by furnishing the upper cylinder with two holes in the side—a 2-inch observation hole with its center 4 inches from the top of the cylinder (excluding cover plate), and a $\frac{3}{4}$-inch thermocouple hole 90° to the right of the observation hole; the bottoms of these two holes are in the same horizontal plane. The interior of the furnace is cylindrical and approximately 7 inches in diameter and 11 inches high. A counterbalanced sheet-iron canopy 24 inches in diameter was connected with a telescopic 8-inch flue to an exhaust system to conduct the hot gases out of the room. The canopy is shown in a raised position in Plate IV, *B*; when in use it is pulled down to the furnace cover.

The joint between the upper and the lower cylinder was made fairly gas tight by spreading on the top of the lower cylinder a thick paste of alundum cement to a depth of one-half inch and then firmly pressing the upper cylinder into place. The outer part of the joint was also smoothed over with alundum cement.

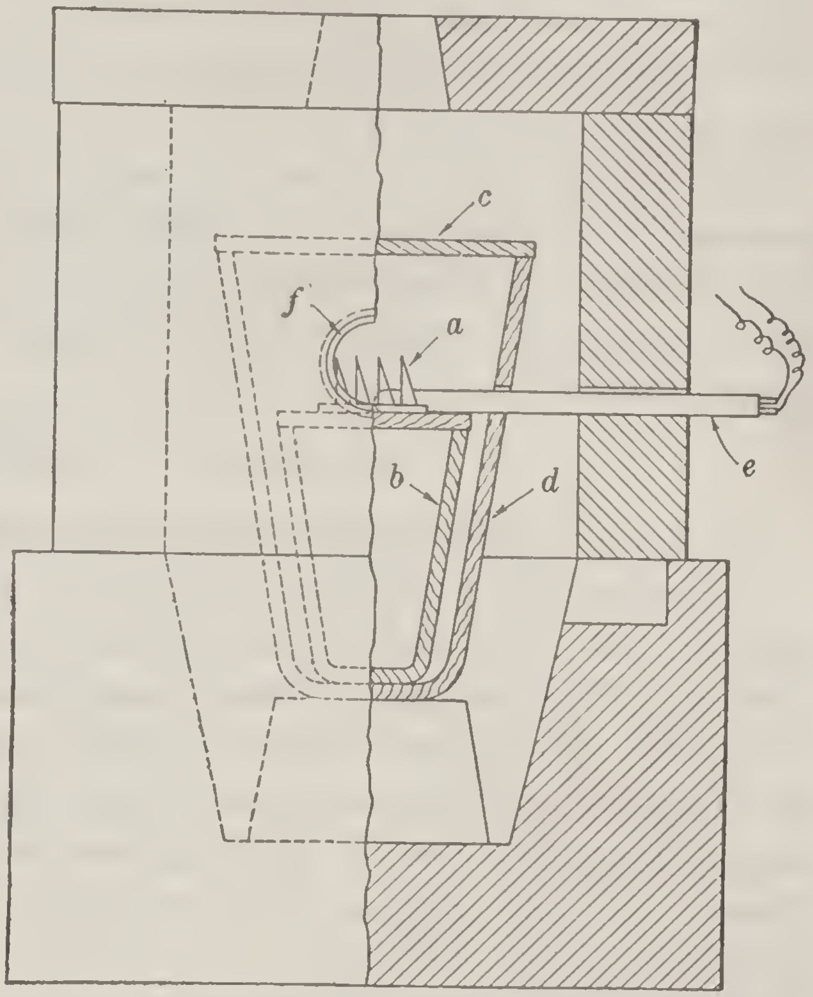

Figure 31.—Section of No. 3 melter's furnace arranged for fusion tests.

The interior of the furnace as arranged for making a test is shown in figure 31. The ash cones *a* are supported on a plate of alundum cement serving as a cover for the crucible *b*, which is 4 inches high and 3 inches in diameter at the bottom, outside dimensions. The cone-supporting crucible is within the crucible *d*, which with the cover *c* serves as a muffle. Crucible *d* is 7 inches high, 5 inches in diameter at the top, and 3 inches in diameter at the bottom, and is provided with observation and thermocouple holes corresponding to those in the furnace cylinder. The platinum and platinum-rhodium thermoelement is protected from the furnace gases by a glazed Marquardt porcelain tube *e* one-fourth inch in diameter. A fused silica tube *f*, 1$\frac{7}{8}$ inches in external diameter and 6 inches long, is placed in the 2-inch observation hole of the furnace, the inner end projecting through the wall of the crucible *d*, and the other end projecting out of the furnace a distance of 2 inches. A brass sleeve carrying a thin glass window was slipped on the outer end of the observation tube to prevent the escape of burning gas, which would have interfered with convenient observation of the cones.

The fire-clay furnace cylinders and cover will withstand a temperature of 1,650° C. It has, however, been difficult to obtain suitable refractory crucibles. Fused-silica ware was first used but was later abandoned on account of high first cost and short life owing to crystallization; alundum ware also went to pieces after a few runs. Finally the ordinary hard burned-clay assay crucibles were tried and found to be suitable, provided they were not heated to a temperature exceeding 1,500° C.

The average life of a crucible under these conditions is six runs; however, they cost only a few cents each, and are quickly replaced. Experiments are being made with various materials in order to obtain a cheap and durable material that may be used for temperatures up to 1,650° C.

METHOD OF OPERATION.

Natural gas, that used by the city of Pittsburgh, Pa., was supplied to the furnace at a pressure of approximately 10 to 12 inches water gage, through a ¾-inch pipe; air was supplied from a Root blower through a 1½-inch pipe. The blower developed a pressure of 2 to 3 pounds to the square inch. As this pressure was inadequate for attaining temperatures above 1,500° C. an ordinary oxygen tank was connected to the air line and the air was enriched with oxygen whereby temperatures up to the melting point of platinum (1,755° C.) were readily obtained.

With stronger air pressure, at least 5 pounds per square inch, temperatures up to 1,700° C. should be easily obtained without recourse to oxygen.

The coal ash was prepared and the cones were mounted in the manner described on pages 29–30.

Although several cones can be heated simultaneously in the gas furnace as shown in figure 31, all tests described in the following experiments were made with single cones, in order to eliminate any possible influence of volatile constituents of one cone on the softening point of another cone. The mounted cone having been placed in proper position in the crucible *d* (fig. 31) with the cover *c* in place, the gas was turned on, ignited, and allowed to burn about 10 minutes before the cover plate of the furnace was replaced; during this time the air and gas was gradually increased so as to slowly heat the crucible without cracking it. After the cover plate was in place on the furnace, the proportion of air was increased sufficiently to cause the combustion to take place just above the tuyères, and yet maintain a flame at least 6 inches high above the opening in the cover plate. From this point the temperature was gradually increased by a suitable adjustment of gas and air until it was at least 200° C. below the probable down point of the cone; then a temperature increase of 5° C. per minute was started and maintained until deformation commenced. When the cone started to bend the rate of heat increase was reduced to 2° C. per minute, and this rate was maintained until the end of the test. During the entire period of heating a reducing atmosphere was maintained in the furnace by using the minimum amount of air necessary for attaining the desired temperatures. A luminous reducing flame issued from the opening of the cover plate to a height of at least 6 inches, except at the higher temperatures above 1,450° C., which could not be attained without a larger proportion of air to gas. Fortunately a strongly reducing atmosphere is not so essential at the higher temperatures, as refractory ashes, owing to their low content of iron oxide, are only slightly affected by oxidizing or reducing atmospheres.

COMPOSITION OF ATMOSPHERE SURROUNDING CONES.

While the furnace was operated under the conditions described above, samples of gas were taken from the interior of the muffle crucible *d* (fig. 31) at various temperatures between 800° and 1,500° C. The results of analyses of the gas samples, given in Table 19 following, show a fairly uniform composition over the given temperature range. Practically all the oxygen of the air had been consumed.

TABLE 19.—*Results of analyses of gas samples taken at various temperatures from No. 3 melter's furnace.*

Temperature where sample was taken.	Percentage by volume of—							Ratio.	
	CO_2.	CO.	H_2.	O_2.	C_2H_4, etc.	CH_4.	N_2.	$\frac{CO_2}{CO+CO_2}$	$\frac{CO}{CO+CO_2}$
° *C.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
800........	6.1	8.3	7.0	0.0	0.6	1.6	76.4	42	58
900........	6.1	7.9	7.8	.5	.3	1.8	75.6	44	56
1,000.......	6.4	8.5	7.7	0	.2	.4	76.8	43	57
1,100.......	5.9	8.3	8.2	.3	.5	.5	76.3	42	58
1,200.......	6.2	8.5	9.1	.4	.2	.3	75.3	42	58
1,300.......	6.4	8.5	8.6	.1	.1	.3	76.0	43	57
1,400.......	6.9	7.3	6.5	.2	.2	.3	78.6	49	51
1,500.......	9.3	6.6	6.5	.2	.0	.3	77.1	59	41

These analyses do not show the real composition of the furnace atmosphere in that the water vapor was not determined. However, it is possible to calculate the percentage of water vapor from the analysis of the natural gas used and the products of combustion, as follows:

CALCULATION OF PERCENTAGE OF WATER VAPOR IN GAS SAMPLE.

Composition of natural gas used.

Constituent.	Per cent.
Methane (CH_4)	78.7
Ethane (C_2H_6)	20.3
Nitrogen (N_2)	1.0
	100.0

78.7 c.c. CH_4 forms on combustion 78.7 c.c. $CO+CO_2$ and 157.4 c.c. H_2+H_2O.
20.3 c.c. C_2H_6 forms on combustion 40.6 c.c. $CO+CO_2$ and 60.9 c.c. H_2+H_2O.
119.3 218.3

$$\text{Ratio of } H_2+H_2O \text{ to } CO+CO_2 \text{ in products of combustion} = \frac{218.3}{119.3} = 1.83$$

Hence water vapor in products of combustion = $(CO+CO_2)\ 1.83-H_2$.[a]

The percentage of water vapor was calculated in this way for each analysis, and the results were combined with the results of the analysis in order to obtain the actual composition of the furnace atmosphere at various temperatures. The values so found are given in Table 20 following.

TABLE 20.—*Composition of furnace atmosphere at various temperatures.*

Temperature.	Percentage by volume of—								Reducing gases.a	Oxidizing gases.b	Ratio of reducing gases to oxidizing gases.
	N_2.	C_2H_4.	CH_4.	H_2.	CO.	CO_2.	H_2O.	O_2.			
° *C.*											
800......	64.0	0.5	1.3	5.9	6.9	5.1	16.1	0.0	14.6	21.2	41:59
900......	64.3	.3	1.4	6.6	6.7	5.2	15.1	.4	14.0	20.7	40:60
1,000.....	64.1	.2	.4	6.4	7.1	5.4	16.4	.0	14.1	21.8	39:61
1,100.....	64.8	.4	.4	7.0	7.0	5.0	15.1	.3	15.2	20.4	42:58
1,200.....	63.9	.2	.3	7.7	7.2	5.3	15.1	.3	15.4	20.7	43:57
1,300.....	63.9	.1	.3	7.3	7.2	5.4	15.7	.1	14.9	21.2	41:59
1,400.....	65.8	.2	.2	5.4	6.1	5.8	16.3	.2	11.9	22.3	35:65
1,500.....	62.9	.0	.3	5.3	5.3	7.6	18.4	.2	10.9	26.2	29:71

a $C_2H_4+CH_4+H_2+CO$. b $CO_2+H_2O+O_2$.

Methane (CH_4), ethane (C_2H_4), hydrogen (H_2), and carbon monoxide (CO) are reducing gases; oxygen (O_2), carbon dioxide (CO_2), and water vapor (H_2O) are oxidizing gases; nitrogen may be regarded as a neutral gas having neither oxidizing nor reducing properties. On this basis the ratio of reducing to oxidizing gases was fairly constant at 40 to 60 in the temperature range between 800° and 1,300° C.; above 1,300° C. the atmosphere became more oxidizing, the ratio being 29 to 71 at 1,500° C. There is seemingly no danger of obtaining in the gas furnace an atmosphere that is too reducing in character. However, care must be observed to maintain the largest possible excess of gas over air; otherwise the minimum softening point corresponding to a ferrous iron slag will not be obtained.

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[a] Neglecting water vapor in gas and air supplied to the furnace and assuming no other forms of carbon in combustion products.

TESTING FOR MAGNETIC SLAG AND METALLIC IRON.

In connection with the fusion tests made in atmospheres of equal parts of hydrogen and water vapor and of equal parts of carbon dioxide and carbon monoxide, it was shown that ash cones fused in such atmospheres showed little or no magnetism when suspended in the field of a strong electromagnet. This test was therefore used for determining the efficiency of reduction in the gas furnace. The electromagnet (fig. 32) consisted of a rod of Swedish iron, *a*, 6¾ inches long and 1 inch in diameter, upon which were wound 250 feet of No. 18 B. & S. gage insulated copper annunciator wire, *b*, in 10 layers, the whole being covered with electrician's tape to keep the wire in place. The fused cone was placed in the copper stirrup, *c*, which was free to swing on a thread supported from above. A current of 3½ amperes was used to energize the magnet.

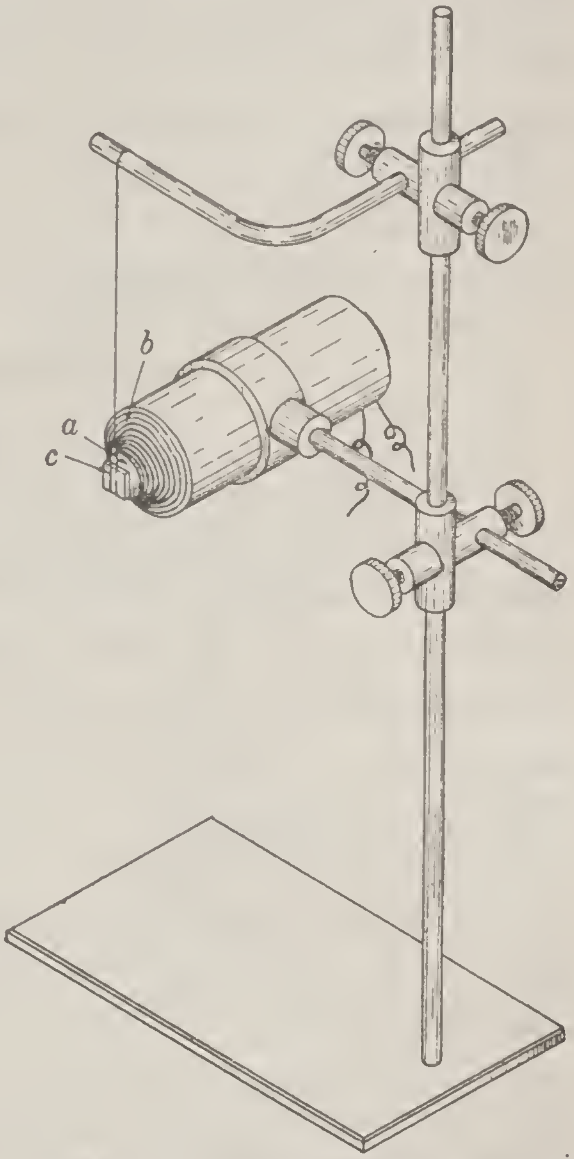

FIGURE 32.—Electromagnet used to determine magnetic properties of fused cones.

A dilute solution of copper sulphate was poured over a freshly ground surface of a fused cone. Metallic iron was then easily detected when this surface was examined under the microscope, as any particles of iron became copper plated.

TEMPERATURE MEASUREMENTS.

Most of the temperature measurements were made with a thermoelement of platinum and platinum-rhodium and a Siemens and Halske high-resistance millivoltmeter. The cold junction was kept at the temperature of melting ice. The thermoelement was protected from the furnace gases by a glazed Marquardt porcelain tube of five-sixteenth inch external diameter. This outfit could be safely used up to temperatures of 1,550° C. For higher temperatures an optical pyrometer of the Holborn-Kurlbaum type was used. The accuracy of both the thermocouple and the optical pyrometer was checked at regular intervals by determining the melting points of gold (1,063° C.), diopside (1,391° C.), and nickel (1,450° C.), as in the earlier experiments.

The whirling flame produced by the three tangential burners heated the muffle crucible so evenly that it was difficult to see the cones and observe their deformation until means was provided for locally cooling the cones slightly below the temperature of the crucible wall. For this purpose a ¼-inch Marquardt porcelain tube was inserted through the openings provided for the thermocouple tube; the tube was so placed that when it was connected with the compressed-air pipe a jet of air was directed against the cones, cooling them momentarily, and thus rendering them visible.

RESULTS OF TESTS.

To furnish a basis for comparison of the results obtainable in the No. 3 melter's furnace with the results of fusion tests with other equipment, determinations were made of the softening temperatures of the same 48 coal ashes that had previously been fused in air, in an atmosphere of equal parts of hydrogen and water vapor, and in an atmosphere of equal parts of carbon monoxide and carbon dioxide. The size of cone and rate of heating was exactly the same in each of the different atmospheres. The results obtained are given in Table 13 (p. 68) and in figures 24 and 25 (pp. 70 and 71). It will be seen that most of the gas-furnace results agree fairly well with those obtained in the CO-CO_2 atmosphere, the average difference for the two series being only 10° C. and the average softening point curve *d* (fig. 24) for the gas furnace almost coinciding with the CO-CO_2 curve *c*. There is, however, a difference, varying from 30° to 70° C., between the average softening point curve *d* and the average softening point curve *b*, representing the results obtained in the atmosphere of hydrogen and water vapor, which produced the lowest softening temperatures.

COMPARISON OF RESULTS OBTAINED IN TWO DIFFERENT TYPES OF GAS FURNACES OPERATED UNDER MAXIMUM REDUCING CONDITIONS.

The experiments described showed that it was possible to obtain duplicate results in softening-temperature determinations made under maximum reducing conditions in the No. 3 melter's furnace.

The next step in investigating the suitability of the gas furnace for use in a standard method was to determine whether comparable results would be obtained in different types of gas furnaces, provided the furnace construction was such as to allow free access of the combustion gases to the ash cones.

As a different type of furnace, a No. 29 Meker muffle furnace was modified as shown in figure 33 by removing the muffle and replacing it with a porous alundum tube, *a*, 1¼ inches in internal diameter, the open front of the furnace around the alundum tube was closed with refractory fire clay, *b*; a thin rectangular plate *c*, 1½ by 3 inches, composed of kaolin and alumina mixed in equal proportions, was placed over the burner and directly under the alundum tube *a*, to spread the flame and cause uniformly distributed heating.

Within the tube *a* were placed two short alundum capsules *d* and *e*, the former with its closed end in front, near which was placed the cone *f*, mounted as usual in a base of kaolin and alumina. Radiation was cut down by placing the capsule *e* in front of the cone; a small opening in the closed end of this capsule permitted observation and measurement of the temperature with an optical pyrometer of the Holborn-Kurlbaum type. The temperature measurements were checked in the usual manner by observing the melting points of pure gold wire and crystals of diopside. The burner connections were so arranged that either natural gas or coal gas could be used, with air at a pressure of 2 pounds per square inch. Owing to the low air pressure it was necessary to enrich the air with oxygen in order to attain the highest temperatures.

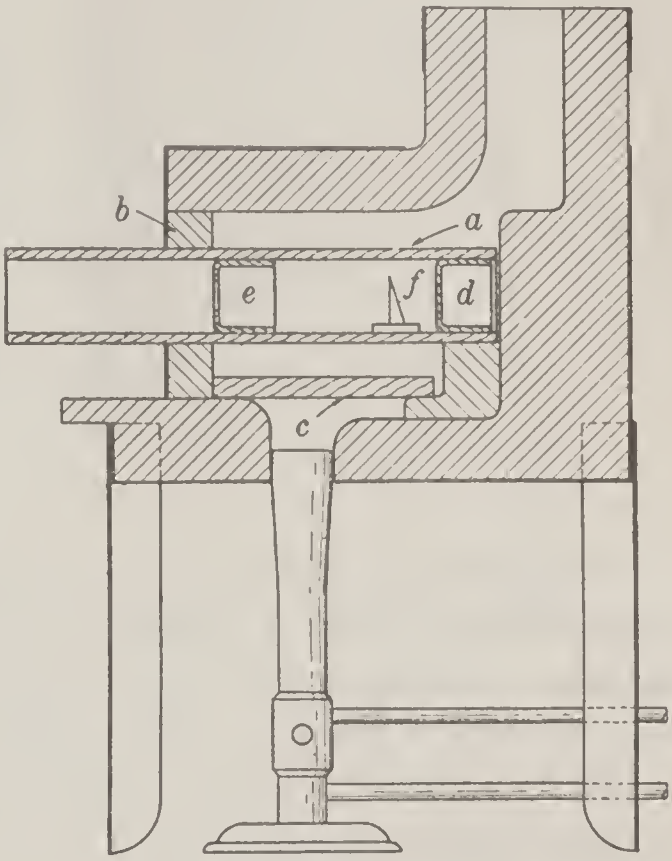

FIGURE 33.—Modified No. 29 Meker furnace.

The method of making and mounting the ash cone, its size, and the rate of heating it were the same as in the tests with the No. 3 gas furnace. The furnace was operated at all times under maximum reducing conditions, a flame issuing from the chimney to a height of approximately 6 inches. Table 21 following shows the results of these tests in comparison with those with the No. 3 melter's furnace.

TABLE 21.—*Results of tests to determine softening temperatures in No. 3 melter's furnace and in modified Meker furnace.*

Ash sample No.	Softening temperatures in—			Differences.	
	No. 3 furnace with natural gas.	Meker furnace with natural gas.	Meker furnace with artificial gas.	2 and 3.	3 and 4.
1	2	3	4	5	6
	°*C.*	°*C.*	°*C.*	°*C.*	°*C.*
86	1,076	1,087	1,074	−11	+13
88	1,125	1,134	1,124	− 9	+10
83	1,145	1,156	1,115	−11	+41
11	1,156	1,165	1,137	− 9	+28
69	1,166	1,179	1,177	−13	+ 2
100	1,172	1,153	1,153	+19	0
90	1,177	1,185	1,184	− 8	+ 1
93	1,198	1,195	1,146	+ 3	+49
17	1,218	1,212	1,189	+ 6	+23
68	1,229	1,234	1,224	− 5	+10
97	1,236	1,266	1,276	−30	−10
76	1,250	1,248	1,219	+ 2	+29
13	1,262	1,288	1,329	−26	−41
18	1,277	1,271	1,224	+ 6	+47
72	1,304	1,318	1,316	−14	+ 2
67	1,316	1,329	1,304	−13	+25
74	1,327	1,295	1,295	+32	0
95	1,325	1,283	1,291	+42	− 8
89	1,325	1,295	1,288	+30	+ 7
91	1,341	1,283	1,318	+58	−35
77	1,401	1,358	1,358	+43	0
Average	....	....	....	+ 4½	+ 9

A representative lot of 21 coal ashes, varying in softening temperatures from 1,070° to 1,400° C., were tested in the two furnaces, the determinations in the No. 3 melter's furnace being made by one of the authors,[a] and the determinations in the Meker furnace being made by another member[b] of the laboratory force. Thus the use of a reducing atmosphere in a gas furnace was put to a more severe test by having different operators as well as different types of furnaces and pyrometers. The agreement in the results obtained, as shown in Table 21 and in figure 34 is even better than was expected and is within the requirements of a standard method. The largest difference was 45° C., whereas 75 per cent of the samples checked within 25° C. The average difference for the series was 4½° C.

COMPARISON OF RESULTS OBTAINED WITH NATURAL AND WITH ARTIFICIAL GAS.

The tests described were all made with natural gas as the furnace fuel. The results of tests with artificial gas shown in the table and in figure 34 are explained later. The combustible constituents of the

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[a] A. E. Hall, assistant chemist.

[b] H. A. Depew, junior physical chemist.

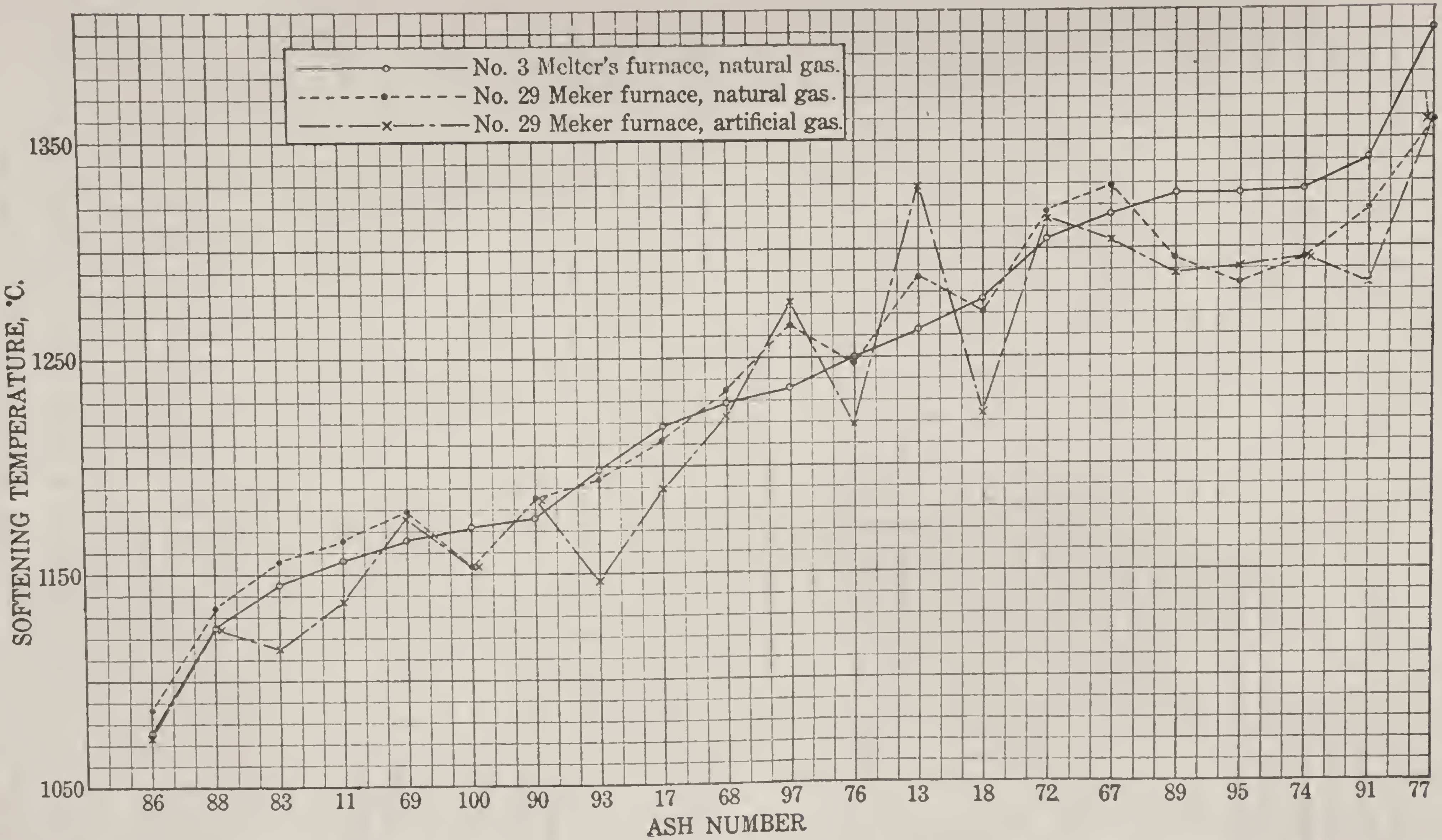

FIGURE 34.—Curves showing softening temperatures determined in No. 3 gas furnace and in modified No. 29, Meker furnace.

natural gas used in the city of Pittsburgh, Pa., comprise paraffin hydrocarbons only, the composition being as follows:[a]

Analysis of natural gas used at Pittsburgh, Pa.

Constituent.	Per cent.
Nitrogen (N_2)	1. 6
Methane (CH_4)	84. 7
Ethane (C_2H_6)	9. 4
Propane (C_3H_8)	3. 0
Butane (C_4H_{10})	1. 3
	100. 0

In order to determine whether the composition of the fuel gas used had any influence on the softening temperature of an ash when heated under reducing conditions, the softening points of the same lot of ash samples were again determined in the Meker furnace under exactly the same conditions as before except that artificial gas was used instead of natural gas. The artificial gas used in Pittsburgh is made up of approximately 1 part of carburetted water gas and 3 parts of coal gas. Burrell, Seibert, and Robertson[b] give the following complete analysis for this gas:

Analysis of Pittsburgh artificial gas.

Constituent.	Formula.	Per cent.
Carbon dioxide	(CO_2)	2. 63
Oxygen	(O_2)	. 81
Carbon monoxide	(CO)	13. 25
Hydrogen	(H_2)	37. 33
Methane	(CH_4)	31. 13
Ethane	(C_2H_6)	2. 10
Propane	(C_3H_8)	. 43
Ethylene	(C_2H_4)	6. 05
Propylene	(C_3H_6)	. 60
Butylene	(C_4H_8)	. 11
Benzene	(C_6H_6)	1. 33
Nitrogen	(N_2)	4. 23
		100. 00

The results obtained with the artificial gas have been given in Table 21 and in figure 34. The determinations in artificial gas averaged 10° C. lower than those in natural gas. The largest deviation was 49° C., and only 5 samples in a total of 21 deviated more than 30° C. These differences are little greater than would be expected in a parallel series of softening temperature determinations in which the same gas was used; hence the use of natural gas in one laboratory and artificial gas in another should not interpose any obstacle in the way of obtaining closely similar results.

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[a] Burrell, G. A., Seibert, F. M., and Robertson, I. W., Analysis of natural gas and illuminating gas by fractional distillation at low temperatures and pressures: Tech. Paper 104, Bureau of Mines, 1915, p. 16.

[b] Same work, p. 16 and p. 28.

IGNITION OF ASH IN OXYGEN.

The effect of unburned carbon in the ash in causing puffing or intumescence of the melting ash has been discussed, especially with reference to softening tests made in an atmosphere of hydrogen and water vapor. In the H_2–H_2O atmosphere many of the cones had a marked tendency to swell even when the ash had been repeatedly ground and reignited in air and was finally made up into cones without the use of any carbonaceous binding material. Experiments made with the addition of small amounts of sugar carbon or dextrin did not produce any appreciable difference in the softening temperature or swelling tendencies of the ash; hence it was concluded that under the slow rates of temperature increase (2° C. per minute) used, the carbonaceous material reacted with the atmosphere and was removed from the system before the ash began to soften, and that the swelling in H_2–H_2O atmospheres was due to causes other than the presence of carbon.

In CO–CO_2 atmospheres and in the reducing atmosphere of the gas furnace, swelling of the cones was comparatively infrequent. However, a particularly troublesome ash in this respect led to igniting the ash in a current of pure oxygen for several hours at a temperature of 850° C. This treatment practically eliminated swelling of the cone in the reducing atmosphere of the gas furnace, and greatly reduced the degree of swelling when the ash was fused in the H_2–H_2O atmosphere. As swelling of the cone makes the softening point less distinct, and practically prevents the determination of the fluid temperature, it was deemed advisable to regrind the ashes and subject them to a second ignition in oxygen for a period of two hours, at a temperature of 800° to 850° C.

This ignition was conveniently made in the improvised electric furnace shown in figure 35. It consisted of a fused-silica tube *a*, 2¾ inches in internal diameter and 18 inches long, and having a wall one-fourth inch thick. It was open at one end and drawn down at the other end to a tube *b*, of three-eighths inch external diameter. On this tub was wound 90 feet of nichrome ribbon *c*, one-eighth inch wide and 0.032 inch thick (No. 20 B. & S. gage), the turns being as close as possible without touching; the winding was covered with a coating of alundum cement. The silica tube was supported at each end by asbestos (transite) board three-fourths inch thick, which was cut to fit snugly into the sheet-iron shell of the furnace. The space between the silica tube and the shell was packed with kieselguhr (infusorial earth). The sheet-iron cylindrical jacket was 10 inches in diameter and 17½ inches long. Oxygen was supplied from a tank through a rubber tube and a bubbling bottle at the rate of 3 to 5 bubbles per second, the rubber tube being connected to the silica tube at *b*.

Before the ash was ignited it was ground for 20 or 30 minutes in a mechanical agate mortar grinder, which reduced the ash to 200 mesh and finer. Five to ten grams of this finely ground ash was placed in a silica or porcelain capsule *d* 1¾ inches in diameter and five-eighths inch deep, and was transferred to the furnace. Six of these could be placed in the furnace at one time. The open end of the furnace was closed with a snugly fitting plug of "transite." The thermocouple was inserted through a hole in this plug. After one or two runs with the thermocouple, the temperature could be estimated with sufficient accuracy by the eye.

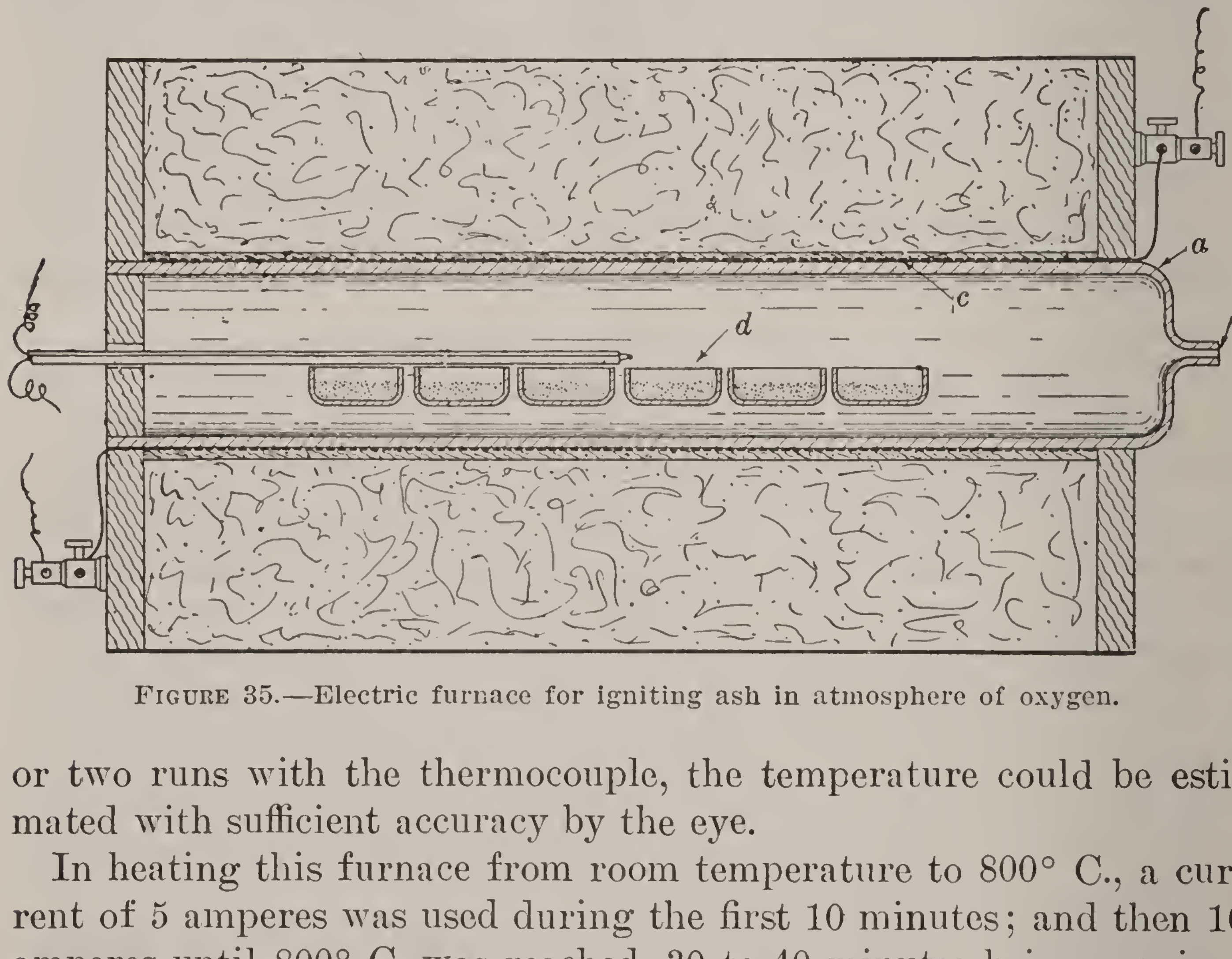

FIGURE 35.—Electric furnace for igniting ash in atmosphere of oxygen.

In heating this furnace from room temperature to 800° C., a current of 5 amperes was used during the first 10 minutes; and then 10 amperes until 800° C. was reached, 30 to 40 minutes being required in all. Five to six amperes was required to maintain the temperature at 800° C. The maximum voltage was 110 volts.

This method of ignition was used in all subsequent experiments.

INFLUENCE OF SIZE AND SHAPE OF CONE.

The ordinary Seger cone used in the ceramic industries is so proportioned that on softening it describes a smooth bend or half circle, the tip touching the base. In the early part of this investigation it was found that similar cones made of molded coal ash did not, as a rule, bend as uniformly as the standard Seger cones. Certain ashes had a decided tendency to flow down into a somewhat spherical lump without bending. Inclination of the cone did not solve the difficulty but, on the contrary, introduced more variation in the results, owing to shrinking and warping of the cone as the sintering temperature was approached. The most uniform results appeared to be obtained with a rather slender cone placed in a vertical position. A cone 1 inch high, having an equilateral triangular base, each side of which was three-sixteenths inch long, was adopted as a tentative standard and used in most of the tests previously described.

On extending this investigation to a wide variety of coal ashes some objections to the slender form of cone became evident, in that the softening-temperature determinations of certain types of ashes could not be duplicated within the usual limits (30° C.). These differences were due to irregularities in the manner of deformation. Some cones flowed or "slumped" down into a lump without bending; other cones bent over in the usual manner. The down point of such cones was always lower, sometimes as much as 200° C. Previously described experiments (pp. 79–86) have shown the impossibility of making the cones composed of certain types of ashes describe a smooth bend on softening even when the cones are mounted in a horizontal position. Furthermore, the deformation temperature of inclined or horizontally mounted cones represents in many instances a comparatively incomplete state of fusion. The authors believe that serious clinker trouble in a furnace corresponds to a more complete fusion of ash, a state in which the resulting slag flows from the force of the gravity alone, without the aid of a bending moment. Reasoning from this point of view it was decided to use a cone with a wider base in proportion to the altitude so that the vertically mounted cone would never bend but always flow or "slump" down into a spherical lump, the temperature at which the cone attained this condition to be taken as the "deformation point" or "softening temperature."

In order to determine the best proportions of base to altitude, 12 ashes, whose softening temperatures ranged from 1,050° to 1,550° C., were molded into cones of the following sizes: $\frac{3}{16}$ by ½ inch, $\frac{3}{16}$ by ⅝ inch, $\frac{3}{16}$ by 1 inch, and ¼ by ¾ inch. These ashes were selected as far as possible from those that had previously given trouble in irregular deformation. The cones were prepared in the usual manner and one of each size was mounted on the same kind of refractory base, and was placed in the No. 3 gas furnace in the position shown in figure 31. An exploration of the interior of the muffle crucible with the thermocouple showed a maximum variation in temperature of only 5° C.; hence each of the four different sizes of cones of a given ash was subected to the same heat treatment, and any variation of results must have been due only to differences in the size and shape of the cones. The furnace was operated with a maximum reducing flame. From a temperature of 800° C. the rate of heat increase was 5° C. per minute until initial deformation of any cone was noticed; the rate of heat increase was then 2° C. per minute until the last cone had either bent with the tip touching the base, or, failing to bend, had collapsed into a spherical lump. The temperature at which each cone reached one or the other of these conditions was recorded as the softening temperature. All the determinations were made in duplicate in order to obtain some data on the tendency toward variation in the manner of deformation of the several different sizes of cones.

RESULTS OF EXPERIMENTS.

In nearly all the tests the slender cone, 1 by $\frac{3}{16}$ inch, described the usual Seger-cone bend, whereas practically each of the other cones settled down into a somewhat spherical lump without bending. The results of the duplicate determinations for each size of cone, which are given in Table 22 following, show that the slender cones, which bent in the tests, were more susceptible to variable results than the cones with wider bases, which softened to a lump, the average difference in results of duplicate tests of a cone $\frac{3}{16}$ by 1 inch being 18.5° C., as compared to 6°, 6.5°, and 8° C., for the nonbending cones.

TABLE 22.—*Results of tests to determine softening temperatures in No. 3 melter's furnace with 4 different sizes of cones.*

[Results in centigrade degrees.]

Ash sample No.	No. 1 cones, 1 inch by $\frac{3}{16}$ inches.			No. 2 cones, $\frac{3}{4}$ by $\frac{1}{4}$ inch.			No. 3 cones, $\frac{5}{8}$ by $\frac{3}{16}$ inch.			No. 4 cones, $\frac{1}{2}$ by $\frac{3}{16}$ inch.			Difference, No. 1 cones and average for No. 2 and No. 3 cones.
	Softening temperature.	Average.	Difference.	Softening temperature.	Average.	Difference.	Softening temperature.	Average.	Difference.	Softening temperature.	Average.	Difference.	
88	1,111 1,124	1,117	13	1,122 1,124	1,123	2	1,124 1,124	1,124	0	1,124 1,124	1,124	0	− 7
104	1,062 1,072	1,067	10	1,132 1,142	1,137	10	1,132 1.132	1,132	0	1,101 1,132	1,116	31	− 68
105	1,190 1,195	1,193	5	1,195 1,195	1.195	0	1,195 1,195	1,195	0	1,195 1,195	1,195	0	− 2
108	1,163 1,169	1,166	6	1,201 1,236	1,218	35	1,211 1,236	1,223	25	1,201 1,206	1,204	5	− 55
107	1,236 1,257	1,247	21	1,246 1,257	1,252	11	1,246 1,257	1,252	11	1,236 1.236	1,236	0	− 5
80	1,153 1,201	1,177	48	1,331 1,342	1,337	11	1,331 1,342	1,337	11	1,331 1,342	1,337	11	−160
67	1,298 1,356	1,327	58	1,353 1,361	1,357	8	1,353 1,361	1,357	8	1,353 1,356	1,355	3	− 30
74	1,321 1,353	1,337	32	1,353 1,364	1,359	11	1,353 1,364	1,359	11	1,353 1,364	1,359	11	− 22
109	1,391 1,407	1,399	16	1,407 1.407	1,407	0	1,407 1,407	1,407	0	1,407 1,407	1,407	0	− 8
110	1,496 1,496	1,496	0	1,505 1,507	1,506	2	1,505 1,507	1.506	2	1,496 1,507	1,502	11	− 10
75	1,496 1,501	1,499	5	1,507 1,512	1,510	5	1,507 1,512	1,510	5	1,507 1,512	1,510	5	− 11
106	1,518 1,531	1,525	13	1,531 1,531	1,531	0	1,531 1,531	1,531	0	1,531 1,531	1,531	0	− 6
Average	....	....	19	....	....	8	....	....	6	....	....	6½	− 32

The effect of varying the size and the shape of the cones on the deformation points is clearly shown in figure 36, in which the average softening temperature of each ash and of cones of each different size is plotted. The results for the No. 2 and No. 3 cones, which had the same ratios of base to altitude, are always within 10 degrees of each other. The results for the No. 4 cone, the shortest tried, also agree fairly well with those for the No. 2 and No. 3 cones. All of these cones deformed to a lump without bending. On the other hand, all of the No. 1 cones, the slender cones that usually bent on softening, had a lower softening point than the cones of the other three sizes, this difference ranging from 2° to 160° C. The observed softening point of a cone made from ash No. 80 may vary from 1,180° to 1,340° C., depending on whether the cone starts to bend at the incipient softening or whether it flows down to a lump without bending. As such variations are likely to occur with bending cones,

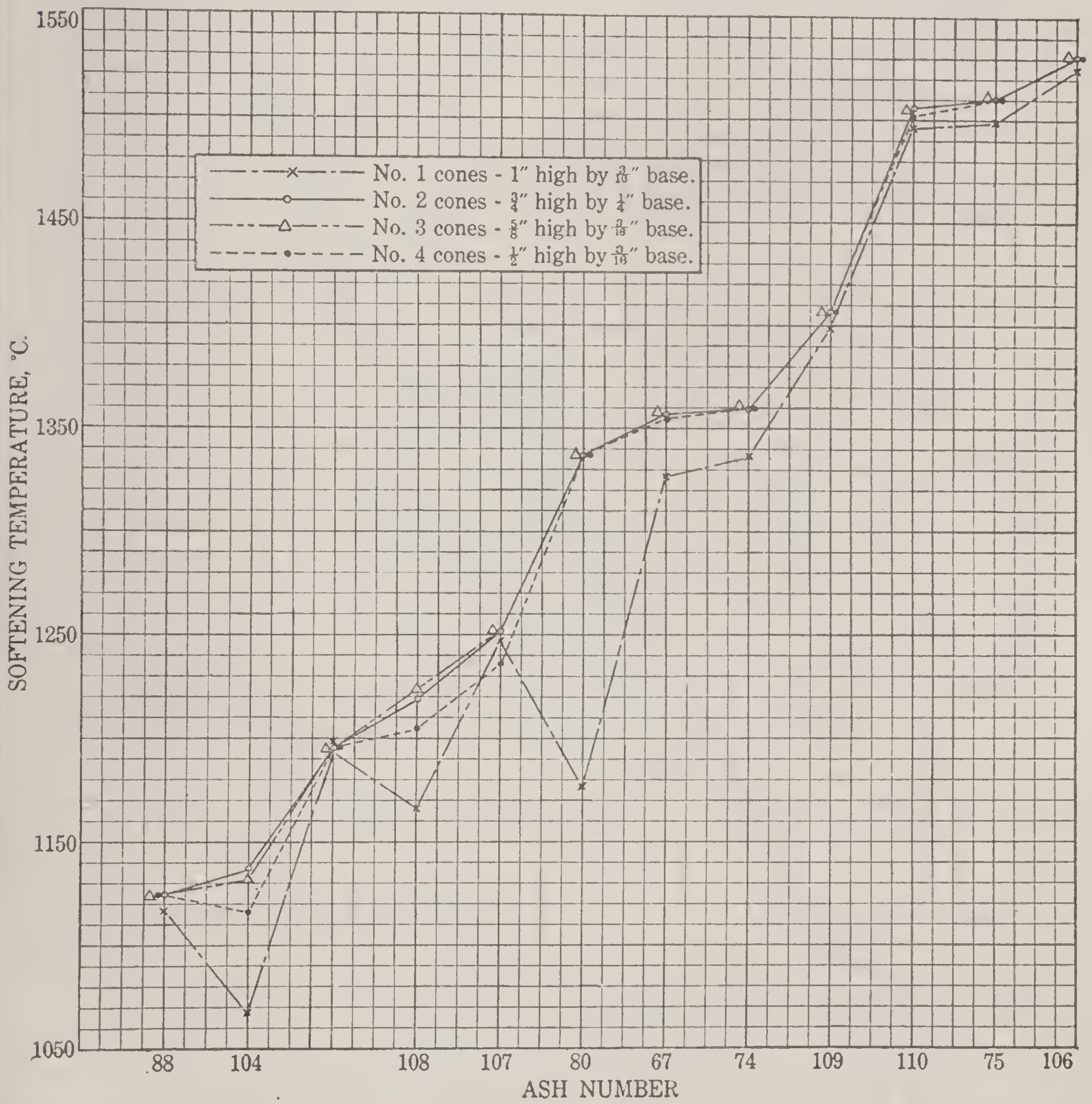

FIGURE 36.—Curves showing softening temperatures obtained in No. 3 melter's furnace with four different sizes of cones.

it seems best to rule out entirely this type of test piece and depend on one so proportioned that the force of gravity and surface tension will change the form from that of a pyramid or cone to that of an approximate sphere. Although perhaps the temperature at which this transition takes place can not be quite so definitely fixed as can the temperature at which the tip of a bending cone touches the base, the change is certainly less subject to large variations due to irregular bending, and in the opinion of the authors more nearly represents the state of fusion of ash when it forms troublesome clinkers.

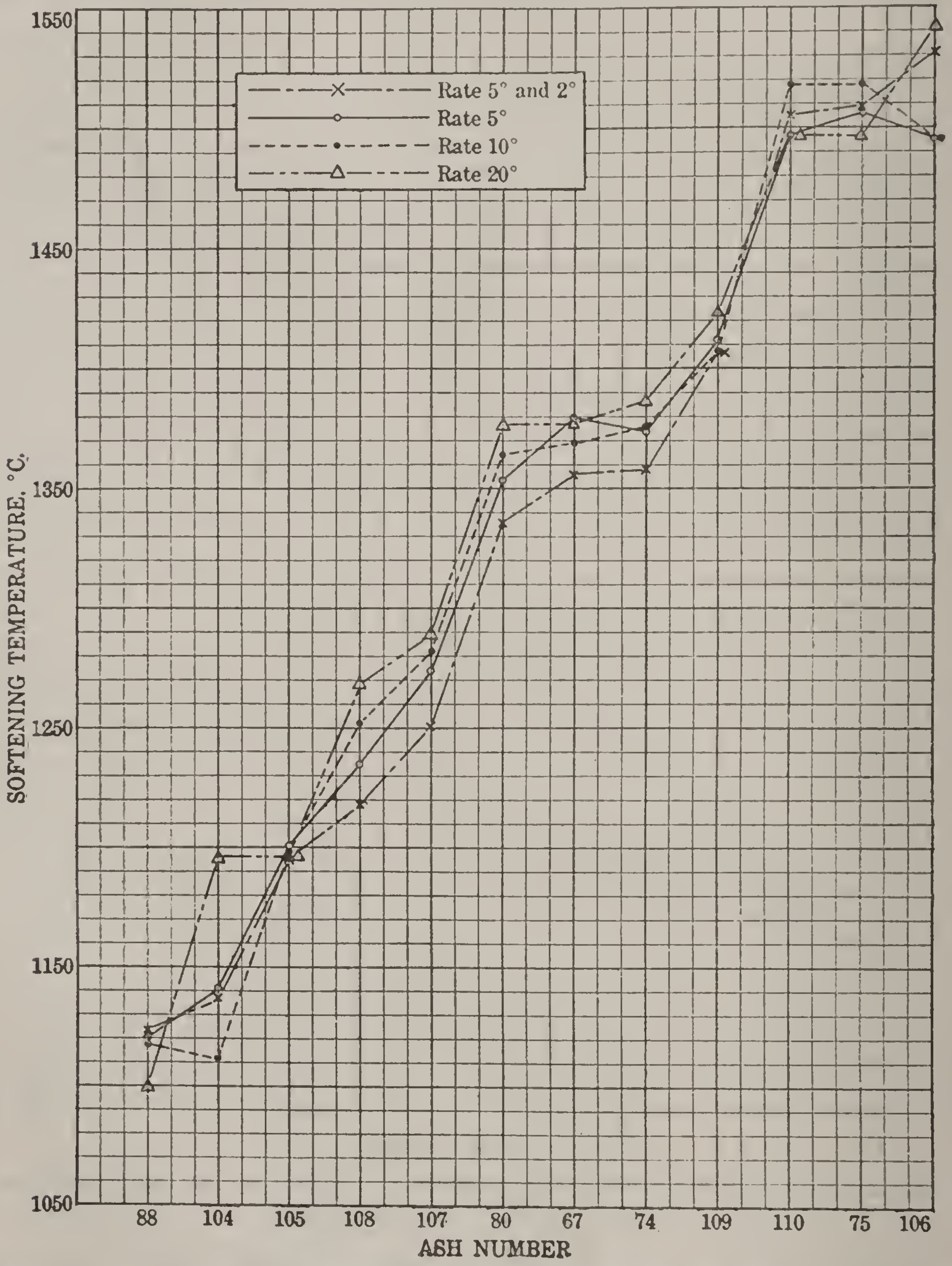

FIGURE 37.—Curves showing softening temperatures obtained in No. 3 melter's furnace at four different rates of heating.

As a result of the experiments described, a standard size of cone having an altitude of three-fourths inch, and a triangular base of one-fourth inch was used in the subsequent experiments. The softening temperature was taken at the point where the cone deformed to a somewhat spherical lump, as shown in Nos. 2, 3, and 4 cones in Plate III, *B* (p. 66).

INFLUENCE OF RATE OF HEATING.

Based on the results of experiments to determine the influence of rate of heating in oxidizing atmospheres of air and in reducing atmospheres of hydrogen, a temperature increase of 5° C. per minute to initial deformation, and then an increase of 2° C. per minute to final deformation was adopted as a standard rate in the experimental work so far described. In view of the importance of making the time of conducting a test as short as possible in commercial testing it seemed desirable to repeat these rate experiments under the new conditions prevailing in the gas furnace when operated under maximum reducing conditions.

The same 12 samples of coal ashes used in the "size-of-cone" experiments were molded into cones ¼ by ¾ inch and heated to the softening point in the No. 3 melter's furnace at four different rates, as follows: 5° C. per minute until initial softening, then 2° C. per minute, 5° C. per minute, 10° C. per minute, and 20° C. per minute. The results obtained are given in Table 23 and in figure 37. In general, the lowest rate of heating produced the lowest softening temperature. With only one exception the results of the 5° and 10° rates were within 20° of each other, and this exception deviated only 30°. It is therefore recommended that the rate of heating be kept between these limits. As a number of cones are usually heated simultaneously, it is advisable to start with a temperature of 800° C.

TABLE 23.—*Results of tests to determine softening temperatures in No. 3 melter's furnace at four different rates of heating.*

Ash sample No.	Rate of temperature increase per minute, ° C.				Differences, ° C.					
	5 and 2[a].	5	10	20	5 and 2 and 5.	5 and 2 and 10.	5 and 2 and 20.	5 and 10.	5 and 20.	10 and 20.
38	1,123	1,120	1,118	1,072 *1,127* 1,099	+3	+5	+24	+2	+21	+19
104	1,137	1,141	1,112	1,189 *1,204* 1,196	−4	+25	−59	+29	−55	−84
105	1,195	1,201	1,199	1,196	−6	−4	−1	+2	+5	+3
108	1,218	1,235	1,252	1,268	−17	−34	−50	−17	−33	−16
107	1,251	1,273	1,282	1,289	−22	−31	−38	−9	−16	−7
30	1,336	1,353	1,364	1,377	−17	−28	−41	−11	−24	−13
57	1,357	1,379	1,369	1,377	−22	−12	−20	+10	+2	−8
74	1,358	1,373	1,375	1,386	−15	−17	−28	−2	−13	−11
109	1,407	1,412	1,407	1,423	−7	0	−16	+5	−11	−16
110	1,506	1,496	1,518	1,497	+10	−12	+9	−22	−1	+21
75	1,509	1,507	1,518	1,497	+2	−9	+12	−11	+10	+21
106	1,531	1,496	1,496	1,542	+35	+35	−11	0	−46	−46
Average					−5	−7	−18	−2	−13½	−11½

[a] 5° C. per minute until initial deformation of cone; then 2° C. per minute.

DETERMINATION OF FLUID TEMPERATURE.

That the softening temperature of an ash, without any other information as to the change of viscosity of the mixture with temperature, must necessarily be a rather inadequate indication of probable clinkering properties has been generally recognized by most investigators who have studied this subject. Some ashes melt to a comparatively fluid slag in a short temperature range, 10° to 20° C.; others soften very slowly, there being an interval of 50° to 150° C.

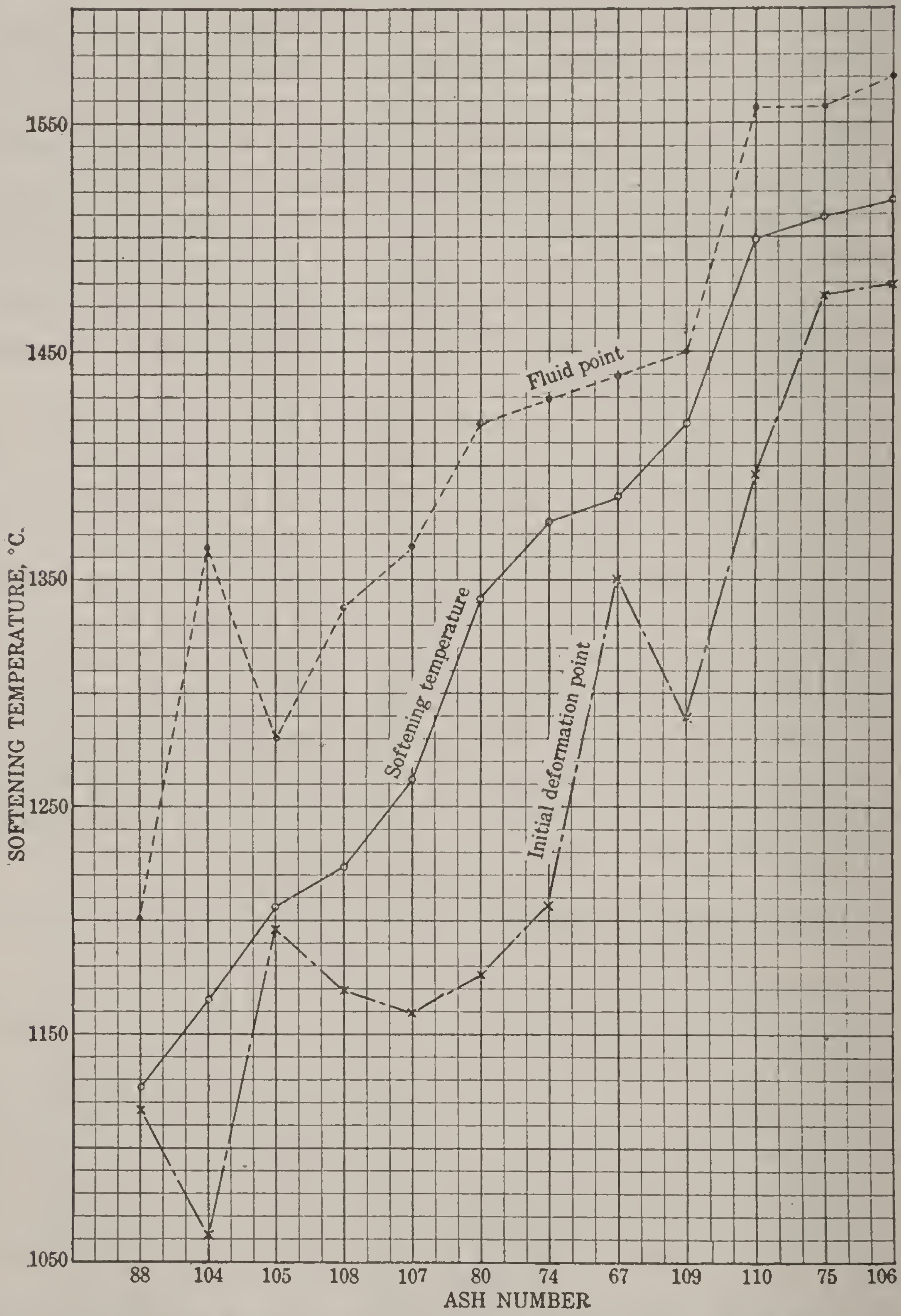

FIGURE 38.—Curves showing temperatures of initial deformation and softening and fluid temperatures of 12 ashes in No. 3 melter's furnace.

between the initial deformation of the cone and its final softening point. This temperature range has been termed by the authors the "softening interval." It is believed that the length of the softening interval, when considered with the softening point, will ultimately be proved to have some connection to clinker formation. Furthermore, at the softening temperature the fusing ash is yet in a pasty and viscous condition; the fusion process has not reached the point where the slag is fluid enough to run and drip through the grate bars of a furnace, as sometimes happens in particularly troublesome clinker formation.

Therefore it is desirable by some simple method to determine the approximate fluid temperature of an ash. One method is to continue heating the cone, after the softening point is reached, to the temperature at which the spherical lump of slag becomes fluid and spreads out in a thin layer over the base in the manner of cone 5 in Plate III, *B* (p. 66). On trying this experiment with the same 12 samples of ashes used in the preceding experiments, it was found entirely practicable to determine this approximate fluid temperature. It was, of course, impossible to reproduce these fluid temperatures in duplicate tests as closely as the softening temperatures, as the end point was less definite than the point at which the cone became a spherical lump. The fluid point, however, was sufficiently defined to have some significance. For example, referring to figure 38, in which the results of these tests are plotted, ash sample 104 began to deform at 1,060° C., it reached the softening temperature at 1,165° C., and became fluid at 1,365° C.; the total temperature interval from initial deformation to fluid point was 305° C. On the other hand, ash sample 88, which began to deform at a higher temperature—1,117° C.—became fluid at a much lower temperature, namely, 1,200° C., the total temperature interval being only 83° C. This would appear to represent the type of ash that might cause troublesome clinkers; that is, the slag would probably spread out over the grate bars in the furnace at a comparatively low temperature, choking off the draft and interfering with combustion to a much greater degree than would ash sample 104. All the "critical points"—initial deformation point, softening temperature, and fluid point—should therefore be taken into consideration in judging the probable clinkering properties of an ash. These three points give approximate data on the change in viscosity with temperature, a change that varies greatly in different ashes. All the results obtained in this series of tests are given in Table 24 following.

TABLE 24.—*Results of tests to determine initial-deformation, softening, and fluid temperatures of 12 samples of ashes in No. 3 melter's furnace.*

[Rate of temperature increase, 5° C. per minute; cones $\frac{3}{4}$ by $\frac{1}{4}$ inch; values in table in centigrade degrees.]

Ash sample No.	Critical points.			Fusion range.		
	Initial-deformation temperature.	Softening temperature.	Fluid temperature.	Start to down point, or softening interval.	Down point to fluid, or flowing interval.	Start to fluid, or total fusion interval.
88	1,117	1,127	1,201	10	74	84
104	1,062	1,165	1,364	103	199	302
105	1,196	1,206	1,280	10	74	84
108	1,169	1,223	1,337	54	114	168
107	1,159	1,262	1,364	103	102	205
80	1,176	1,342	1,418	166	76	242
74	1,206	1,375	1,429	169	54	223
67	1,350	1,386	1,439	36	53	89
109	1,289	1,418	1,450	129	32	161
110	1,396	1,499	1,557	103	58	161
75	1,474	1,509	1,557	35	48	83
106	1,479	1,516	[a] 1,570	37	54	91

[a] Estimated above 1,557° C.

DISCUSSION OF RESULTS OBTAINED IN GAS FURNACE WITH REDUCING ATMOSPHERE.

It was found possible to obtain consistent check results when the No. 3 melter's furnace and the modified Meker furnace were operated under maximum reducing conditions. Such conditions were obtained by allowing the combustion gases to circulate freely about the ash cones, and by so regulating the proportions of gas to air that a reducing flame at least 6 inches high burned at the outlet of the furnace. The softening temperatures thus obtained were generally between the values obtained in atmospheres of equal parts of H_2 and H_2O and atmospheres of equal parts of CO and CO_2. Usually they were very close to the results obtained in the $CO-CO_2$ atmospheres. Most of the iron oxide was reduced to the ferrous form, as shown by the nonmagnetic or only slightly magnetic nature of the fused cones. No metallic iron was found. A given ash sample showed practically the same softening temperature; whether natural or artificial gas was used in the furnace made no material difference. Analysis of the products of combustion from the interior of the furnace showed a ratio by volume of 40 per cent reducing to 60 per cent oxidizing gases, which is within the limits found necessary for a minimum softening temperature in $CO-CO_2$ or H_2-H_2O mixtures, and also approximates the composition of the atmosphere that prevails in the fuel bed of a furnace, as shown by the work of Kreisinger, Ovitz, and Augustine.[a]

In view of these facts, the method of determining the fusibility of coal ash, in a gas furnace having a reducing atmosphere, may be accepted as answering the theoretical requirements of a standard method. A furnace of the type of the No. 3 melter's furnace used in the tests was found to possess the following practical advantages for use in commercial laboratories: (1) It is a standard type of furnace obtainable on the market at a low cost; (2) it is easily operated and repaired with the facilities available in the ordinary laboratory of a commercial plant; (3) owing to the tangential burners and swirling flame the required reducing atmosphere and a very uniform temperature can be maintained around the ash cones; (4) either a thermocouple or an optical pyrometer can be used (the thermocouple must, of course, be protected by a glazed Marquardt porcelain tube, and its use is limited to temperatures under 1,550° C.); (5) the fusibility of six different samples may be determined simultaneously, thus greatly decreasing the cost of a determination.

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[a] Kreisinger, Henry, Ovitz, F. K., and Augustine, C. E., Combustion in the fuel bed of hand-fired furnaces: Tech. Paper 137, Bureau of Mines, 1916, p. 61.

RECOMMENDED PROCEDURE FOR STANDARD GAS-FURNACE METHOD.

PREPARING THE ASH.

Fifty to one hundred grams of 60-mesh coal is spread out on a 6-inch fire-clay roasting dish, and completely converted to ash in a muffle furnace at a temperature of 800° to 900° C. Five to ten grams of this ash is transferred to an agate mortar[a] and ground to a fineness of 200 mesh. The ash is then placed in a silica or porcelain capsule, five-eighths of an inch deep and 1¾ inches in diameter, and ignited for a period of two hours in a current of oxygen, at a temperature of 800° to 850° C. This ignition is made to insure complete and uniform oxidation of the ash.

PREPARATION OF CONES.

The ignited ash is moistened with distilled water or 10 per cent dextrin solution and is worked into a plastic mass with a spatula or pestle. If the ash is finely ground a skilled operator can mold and mount cones without using any binding material; however, the authors were not able to detect any deleterious effect from using a 10 per cent solution of dextrin, provided the mounted cone was ignited at a red heat in an open muffle furnace before the fusion test was made. Cones made with dextrin solution are less fragile.

The plastic material is molded into small triangular pyramids three-fourths of an inch high and one-fourth of an inch wide at the side of the base. The pyramids are made by firmly pressing the plastic material with a steel spatula into a brass mold of the dimensions mentioned, the mold being similar to that shown in figure 7 (p. 29). The surface is then struck off smooth and the cone removed from the mold by applying a small knife blade at the base. By lubricating the mold at the start with a little vaseline the cone can be removed at once without waiting for it to dry. When the cones are dry they are mounted in a refractory base composed of equal parts of kaolin and alumina. The base mixture is moistened to make it workable, and a part of it is spread out on a sheet-iron plate. The cone is then mounted in a vertical position in a small hole made in the base, and base material is put into the hole around the bottom of the cone to fill the crevices and make the cone stand firmly. Usually five or six cones are mounted in one base in the manner shown in Plate III, *B* (p. 66). The sheet-iron plate with the test piece is then dried on a hot plate. If dextrin has been used as binder the cones are ignited at a red heat for 30 minutes in an open muffle furnace to remove the carbonaceous material.

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[a] A mechanical agate-mortar grinder will save time where many determinations are made.

METHOD OF HEATING.

The test piece is placed in the muffle crucible of the melter's furnace in the position shown in figure 31 (p. 91), the loosely fitting cover *c* is placed on the crucible, and the gas is ignited. It is necessary to allow the gas to burn about 10 minutes to heat the furnace parts before the large cover plate of the furnace is replaced; otherwise the flame is apt to blow out. During this time the flow of the gas and air is gradually increased at such a rate as will not cause the crucible to crack. After the cover plate is on the furnace, the volume of gas and air is increased sufficiently to cause the combustion to take place just above the tuyères and yet maintain a yellowish flame at least 6 inches above the opening in the cover plate. While such a flame is maintained above the furnace, the temperature is gradually increased, by a suitable adjustment of gas and air, to 800° C., when the rate of heat increase is slowed down to not less than 5° C. and not more than 10° C. per minute. This rate is maintained until the end of the test. It is also important that the 6-inch reducing flame be maintained at the furnace vent throughout the test, if possible, and at all events the temperature should be kept up to 1,450° C. After a test has been completed the supply of gas and air is turned off gradually to avoid cracking the muffle crucible. The critical points observed in the test are defined as follows:

(1) *The initial deformation temperature.*—The temperature at which the first rounding or bending of the apex of the cone takes place, as shown in cone 1 of Plate III, *B*. Such bending must not be confused with a shrinking or warping of the cone.

(2) *The softening temperature.*—The temperature at which the cone has fused down to a spherical lump as shown in cones 2 and 3 of Plate III, *B*. Cone 4 has almost reached the softening temperature.

(3) *The fluid temperature.*—The temperature at which the cone has spread out over the base in a flat layer, as represented by cone 5 in Plate III, *B*.

In order to avoid the confusion that might arise in an effort to report all of the critical points mentioned it is recommended that only the softening temperature be reported as such, the other two points being given in terms of temperature intervals above and below the softening temperature. For convenience these intervals are reported as follows:

(1) *The softening interval*, which is the difference between the softening temperature and the temperature of initial deformation.

(2) *The flowing interval*, which is the difference between the fluid temperature and the softening temperature.

Following is a specimen report in which these intervals are used:

Ash No. 104.

	° C.
Softening temperature	1, 165
Softening interval	105
Flowing interval	200

Owing to the difficulty of obtaining on the market highly refractory muffle crucibles, commercial laboratories will probably not attempt to carry fusion tests above 1,500° C.; the great majority of coal ashes will soften below this temperature. Any ash that withstands 1,500° C. is in the refractory class and is acceptable as regards fusibility. Within these temperature limits the ordinary Denver fire-clay crucible No. K, 7¼ inches high and 4⅝ inches in diameter at the top may be used. Crucibles should be ordered by the hundred, and the manufacturers should be required to drill two holes in the side of each, an observation hole 2 inches in diameter with its center 2 inches from the top of the crucible and a thermocouple hole three-fourths of an inch in diameter 90° to the right of the observation hole. The bottom of this hole should be in the same horizontal plane as the bottom of the observation hole.

A Denver fire-clay crucible No. E, 4½ inches high and 3 inches in diameter at the top, is suitable for supporting the ash cones in their proper position in the large crucible.

If the maximum temperature limit is placed at 1,500° or 1,650° C., a thermocouple made of platinum and platinum-rhodium, with a high-resistance millivoltmeter or a Northrup pyrovolter, is most convenient and accurate for measuring the temperature. The thermocouple must be protected from the furnace gases with a glazed Marquardt porcelain tube. The outside diameter of this tube need not exceed five-sixteenths of an inch. The outfit should be checked frequently by mounting small pieces of pure gold or nickel wire or foil in place of the cones and observing their melting points in exactly the same manner as the melting points of the cones are observed. With a strong reducing atmosphere, 1,450° C. should be obtained for the melting point of the nickel and 1,063° C. for the gold. The thermocouple and the millivoltmeter should also from time to time be standardized throughout the temperature range for which they are used by a suitably equipped standardizing laboratory such as that of the United States Bureau of Standards.

If it should be desired to determine softening points above 1,500° C., special muffle crucibles of fused silica, magnesia, or corundite will be required, and the temperature measurements will of necessity be made with an optical pyrometer, preferably of the Holborn-Kurlbaum or Wanner type.

SUMMARY AND CONCLUSIONS.

Attention is called to the empirical nature of the "fusing or softening-temperature" test and the necessity of adopting some standard and duplicatable method if it is to be used in specifications for the purchase of coal.

The influence of the more important factors affecting the softening temperatures of the coal ashes tested may be summarized as follows:

Fineness of ash.—Ash ground to an impalpable powder tended to soften at a slightly lower temperature than ash that would pass a 100-mesh screen. The difference averaged 6° C., and in no test exceeded 40° C. Ash pulverized to at least 200 mesh was preferred, as it could be molded into more substantial cones than 100-mesh material.

Inclination of cones.—Mounting the cones with a considerable inclination—35° or 40° from the vertical—caused premature deformation points in some cones, which bent only at or near the base at a comparatively early stage in the fusion process, before the ash had softened to the flowing point. Vertical or nearly vertical cones were more free from this source of variation. Horizontally mounted cones usually gave lower softening points than vertical cones, the difference varying from 0° to nearly 200° C. The average difference for 15 samples was 49° C. In general, horizontal or inclined cones indicated a softening point at too early a stage in the fusion process; hence vertically mounted cones are recommended.

Size and shape of cone.—The deformation of a cone depended to some degree on its size and shape. Within limits of ½ to 2½ inches high the actual size had little influence provided the shape was the same; that is, the ratio between the dimensions of the base and of the altitude. Cones made of coal ash did not bend as uniformly as the standard pyrometric cones. Attempts to modify the ratio of base to altitude in order to obtain a more uniform deformation, like the usual Seger-cone bend, resulted in the selection of a rather slender type of cone—each side of base three-sixteenths of an inch; altitude, 1 inch—as having less tendency to soften to a lump without bending. However, later experience showed that certain ashes when molded into slender cones would bend when only a relatively small part of the ash mixture had fused, thus indicating a premature softening temperature; it was also impossible to prevent some ashes from fusing down to a lump even with the slender type of cone. It was, therefore, decided to eliminate entirely the bending tendency by increasing the width of the base; and to take as the softening temperature the point where the cone had softened and flowed down to a somewhat spherical lump. The optimum size for this purpose was determined experimentally and found to be a cone three-fourths of an inch high and one-fourth of an inch wide at the base. This end point was found reproducable within 30° C., and was practically free from the occasional large variations experienced with the slender bending cones.

Carbonaceous matter in the ash.—Tests in which all traces of carbon were removed and in which the cones were made up exclusively with distilled water, showed no appreciable difference in softening temperature when compared with similar tests in which the same ashes were made into cones with 10 per cent dextrin solution which was not burned out before the cones were inserted in the furnace. These tests were made in atmospheres of air, of hydrogen and water vapor, and of carbon monoxide and carbon dioxide at rates of heating not exceeding 5° C. per minute. However, in certain tests in the gas furnace with a reducing atmosphere, more rapid rates of heating caused swelling and intumescence, which was much less pronounced when the ash was previously ignited in oxygen for two hours. In routine work, ignition of the reground ash in oxygen promotes more definite and reliable softening points, especially when the original ash has been incompletely ignited.

Rate of heating.—In general, slower rates of heating gave lower softening points. Rates slower than 2° C. or faster than 10° C. per minute are not advisable. The effect of variations between these limits in purely oxidizing atmospheres was negligible. In a reducing atmosphere of hydrogen the lowest and most uniform results were obtained with a heat increase of 2° C. per minute; however, a temperature increase of 5° C. per minute until initial deformation of the cone occurred, and then reducing the rate to 2° C. per minute gave practically the same results and saved considerable time in making the test. In the gas furnace with reducing atmosphere, the optimum rate of temperature increase was between 5° and 10° C. per minute. These limits are recommended for a standard rate of heating.

Oxidizing or reducing atmosphere.—The atmosphere in which the ash was heated proved by far the most important factor in causing large variations in the softening temperature. The highest softening points were obtained either in an atmosphere of air (platinum-wire resistance furnace), or in a strongly reducing atmosphere of carbon monoxide (Northrup furnace), which prevented the iron oxide from acting as a fluxing agent by reducing it to metallic iron before the softening of the ash began. The lowest softening temperatures were obtained in those atmospheres of mixed gases in which reduction of ferric oxide proceeded mainly to ferrous oxide, the most active phase in the reduction of iron ore as regards slag formation at lower temperatures. The maximum variation in softening temperatures due to different atmospheres ranged from 134° to 396° C.

Fusibility of ash in various mixtures of hydrogen and water vapor.—The softening temperatures of five different coal ashes were determined in various mixtures of hydrogen and water vapor, ranging from 100 per cent hydrogen to 100 per cent water vapor. These results, plotted in the form of curves, showed that for each of the ashes tested there was a high softening temperature in pure hydrogen on one end owing to reduction of iron oxide to metallic iron; a similar high softening temperature in water vapor or air on the other end owing to the iron oxide remaining for the most part in the form of ferric iron or magnetite; and a somewhat lower softening temperature in the middle part, in an atmosphere ranging from 30 to 70 per cent water vapor, owing to the reduction of most of the iron to the ferrous state, in which it combined to form fusible ferrous silicates.

Fusibility of ash in various mixtures of carbon monoxide and carbon dioxide.—Softening-temperature tests in various mixtures of carbon monoxide and carbon dioxide were also made with four of the ashes that had been tested in mixtures of hydrogen and water vapor. Softening-temperature curves similar to the curves obtained in tests with atmospheres of hydrogen and water vapor were obtained, each having a minimum region of fusion between approximate limits of 75 and 10 per cent carbon monoxide. However, it was found necessary to heat the ash cones more slowly in CO–CO_2 atmospheres to attain approximately the same minimum point as was found in H_2–H_2O atmospheres at faster rates of heating. Even with a slower rate of heating in the former atmosphere the average difference in the minimum softening temperatures (down points) of a series of ashes tested in the two atmospheres was 66.1° C. The maximum difference was 197° C. The results obtained in H_2–H_2O atmospheres were generally lower than in $CO–CO_2$ atmospheres. The initial deformation points were practically the same in $CO–CO_2$ and in $H_2–H_2O$ atmospheres.

Fusibility of ash in the reducing atmosphere of a gas furnace.—It was found possible so to operate two types of gas furnaces, in which the combustion products had free access to the coal ash, that an atmosphere of approximately 60 per cent oxidizing and 40 per cent reducing gases was maintained around the coal ash. Softening temperatures similar to those previously obtained in mixtures containing equal parts of carbon monoxide and carbon dioxide were thus obtained in a much simpler type of furnace. By operating the gas furnace with the maximum excess of gas over air, the desired reduction of ferric to ferrous iron was obtained without the production of any metallic iron.

State of oxidation of iron in ash cones fused in various atmospheres.—Ash cones fused in oxidizing atmospheres of air, carbon dioxide, and water vapor contained 67 to 88 per cent of the total iron in the ferric form; in reducing atmospheres of hydrogen and carbon monoxide, 49 to 78 per cent of the iron was present as metallic iron. In mixtures containing equal parts of H_2 and H_2O or of CO and CO_2 78 to 94 per cent of the iron was present as ferrous iron.

State of oxidation of iron in clinker and slags.—Analyses of clinker slags from two different boiler furnaces and one hand-fired experimental furnace showed that fuel-bed conditions are such as to favor the formation of clinkers containing iron principally in the ferrous state as was found in ash cones fused in mixtures of equal parts of H_2 and H_2O or of CO and CO_2.

Standard method of determining the softening temperature of coal ash.—The method of determining the softening temperature of coal ash in a gas furnace with a reducing atmosphere is recommended as a standard method. The furnace is simple, it can be easily operated, and a number of tests can be made simultaneously. When the furnace is operated under the prescribed reducing conditions reproducable results can be obtained, approximating those of the more definite mixtures of equal proportions of CO and CO_2. A standard furnace design and procedure is recommended.

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57. Percy, John. Metallurgy. London, 1875, pp. 46–86. Discussion of composition and physical properties of slags, mainly silicates.

58. Peters, E. D. Principles of copper smelting. 1907, pp. 339–400. Describes practical study of slags, giving composition, melting temperatures, and formation temperatures of copper slags.

59. Rankin, G. A., and Merwin, H. E. The ternary system CaO–Al_2O_3–MgO. Jour. Am. Chem. Soc., vol. 38, 1916, pp. 568–588. A physico-chemical study in which the method of quenching was used to determine the melting temperatures. Concentration-temperature models and equilibrium diagrams are given.

60. Rankin, G. A., and Wright, F. E. The ternary system CaO–Al_2O_3–SiO_2. Am. Jour. Sci., vol. 39, 1915, pp. 1–79. A summary of the complete investigation of this system, giving tables of melting points of minerals, compounds, and eutectics, with triaxial diagrams and concentration-temperature models.

61. Sélivanov, B. Préparation artificielle, fusibilité et quelques autres propriétes des silicates du systeme $2FeO.SiO_2+2CaO.SiO_2$. Rev. Soc. Russe Métal., 1915, p. 328; Rev. Métall., Extraits, t. 12, 1915, pp. 309–318. Chem. Abs., vol. 10, 1916, p. 1157. Describes preparation, fusibility, and some other properties of silicates of the system $2FeOSiO_2 2CaOSiO_2$. Weighed quantities of FeO, CaO, and SiO_2 were heated in a platinum boat to a certain temperature for a given time, in an atmosphere of nitrogen, and allowed to cool. As a result of heating, the FeO forms Fe and Fe_2O_3 in some degree. No melting temperatures are given in abstract account.

62. Shephard, E. S., and Rankin, G. A. The binary system of alumina with silicia, lime, and magnesia. Am. Jour. Sci., vol. 28, 1909, pp. 293–333. Ztschr. anorg. Chem., Bd. 68, 1910, pp. 370–420. A thermal and microscopic study of the binary systems. Temperature-concentration diagrams are given, showing the melting points of compounds and eutectics.

63. Sosman, R. B. The common refractory oxides. Jour. Ind. and Eng. Chem., vol. 8, 1916, pp. 985–990. Summary of reliable data on melting and inversion temperatures, with diagrams of two and three component systems of the oxides SiO_2, Al_2O_3, MgO, CaO, Fe_2O_3, Fe_3O_4, and FeO.

64. Sosman, R. B., and Merwin, H. E. Preliminary report on the system lime-ferric oxide. Jour. Wash. Acad. Sci., vol. 6, 1916, pp. 532–537. States that Fe_2O_3 dissociates to some degree in all melted mixtures of CaO and Fe_2O_3, especially when the latter is in excess. Mixtures containing less than 50 molecular per cent CaO were not heated above 1,250° C.; only two binary compounds could be found, $CaO.Fe_2O_3$ and $2CaO.Fe_2O_3$. $CaO.Fe_2O_3$ dissociated at 1,216° C., liquefying in part; $2CaO.Fe_2O_3$ dissociated at 1,436° C., liquefying in part.

65. Turner, T. The physical and chemical properties of slags. Jour. Soc. Chem. Ind., vol. 24, 1905, pp. 1142–1149. Presents record of experiments at University of Birmingham with blast-furnace slag, in which thermal method of obtaining cooling curves was used. Concludes that silicate slags are heterogeneous mixtures with no definite melting point to which the phase rule is applicable.

66. Vogt, J. H. L. Die Silicatschmelzlösungen. Christiania, vol. 1, 1903, 161 pp.; vol. 2, 1904, 235 pp. Volume 1 deals with minerals found in silicate fusions with special reference to the dependence of mineral formation upon the composition of the solution. Volume 2 deals with the lowering of the melting points of silicate fusions (comparable to lowering of freezing point in solutions by dissolved substance). A general study of slag and slag minerals from the point of view of the physical chemist.

67. Wallace, R. C. Ueber die binären System des Natriumsilicats mit Lithium-, Magnesium-, Calcium-, Strontium-, und Barium-metasilicat; des Lithium metasilicats mit Kalium-, Magnesium-, Calcium-, Strontium-, und Barium-metasilicat; und über das Dreistoffsystem Na_2O-Al_2O_3-SiO_2. Ztschr. anorg. Chem., Bd. 63, 1909, pp. 1–48. Presents cooling curves obtained by the thermal method; gives conclusions regarding solubility, eutectics, and different crystal phases, and discusses dissociation, decomposition, reduction, and oxidation of substances occurring in, or related to, coal ash and the formation of clinker.

68. Baur, E., and Glaessner, A. Gleichgewichte der Eisenoxyde mit Kohlenoxyd und Kohlensäure. Ztschr. physik. Chem., Bd. 43, 1903, pp. 354–368. Discusses the reduction of Fe_3O_4 to FeO and FeO to Fe by CO. Gives tables and curves of results of study of the following reactions:

$$3FeO+CO_2\rightleftarrows Fe_3O_4+CO$$
$$Fe+CO_2\rightleftarrows FeO+CO$$

See pages 53 and 54, this bulletin.

69. Chaudron, G. Réactions réversibles de l'eau sur le fer et sur l'oxyde ferreux. Compt. rend., t. 159, 1914, pp. 237–239. Gives two series of equilibrium data corresponding to following reactions:

$$Fe+H_2O\rightleftarrows FeO+H_2$$
$$3FeO+H_2O\rightleftarrows Fe_3O_4+H_2$$

See also pages 44 to 46 of this bulletin.

70. Deville, H. St. Claire. Action de l'eau sur le fer et de l'hydrogéne sur l'oxyde de fer. Compt. rend., t. 70, 1870, pp. 1105–1111 and 1201–1207. Presents equilibrium data corresponding to reaction:

$$3Fe+4H_2O\rightleftarrows Fe_3O_4+4H_2$$

71. Falcke, V. Die Reaktionen zwischen Eisenoxydul und Kohle und zwischen Kohlenoxyd und Eisen. Ber. deut. chem. Gesell., Bd. 46, 1913, pp. 743–750. States that addition of amorphous carbon has no effect on the pressure produced when FeO (containing 0.5 per cent C) and Fe are heated together at 530° to 660° C., and that the reaction between FeO and CO is different when the CO is passed over the FeO than when the two are heated together in a closed vessel; in the reaction first mentioned much carbon is deposited; in the second reaction no carbon is deposited.

72. ———. Die Reaktionen zwischen Eisenoxydul und Kohle und zwischen Kohlenoxyd und Eisen. Ztschr. Electrochem., Bd. 22, 1916, pp. 121–133; abstracted in Jour. Chem. Soc., vol. 110, pt. 2, 1916, p. 484. Compares results obtained by author with those of Schenck.

73. Glaser, Ferdinand. Ueber Reduktion von Metalloxyden im Wasserstoffstrom. Ztschr. anorg. Chem., Bd. 36, 1903, pp. 1–35. Treats of reduction of various metallic oxides (including Fe_2O_3) in current of hydrogen.

74. Hilpert, S. Ueber die Reduktion von Eisenoxyd durch Wasserstoff und Kohlenoxyd. Ber. deut. chem. Gesell., Bd. 42, 1909, pp. 4575–4581. Temperatures are given for the beginning of reduction of iron oxide prepared by various methods. States that FeO and Fe_3O_4 in a pure form could not be obtained, each sample showing at once all the stages of reduction. On reducing with CO the products were contaminated with C, seemingly in the form of cementite.

75. ———. Ueber die Sauerstoff-Abgabe des Eisenoxyd, bei hohen Temperaturen. Ber. deut. chem. Gesell., Bd. 42, 1909, pp. 4893–4895. States that results varied according to source and previous treatment of oxide. The reaction took place slowly.

76. Hilpert, S., and Beyer, J. Ueber Eisenoxyduloxyde und Eisenoxydul. Ber. deut. chem. Gesell., Bd. 44, 1911, pp. 1608–1619. Describes a method for preparing pure Fe_3O_4 by heating Fe_2O_3 in H_2 containing a definite amount of water vapor. Attempts to prepare FeO by increasing the amount of water vapor and raising the temperature always gave some Fe, and at 1,100° C. the product still contained 1.5 per cent Fe_2O_3. A method was devised for making comparative measurements of the magnetic properties of the products in powdered form. The presence of solid solutions with Fe_3O_4 was shown.

77. Hofman, H. O., and Mostowitsch, W. The behavior of calcium sulphate at elevated temperatures with some fluxes. Trans. Am. Inst. Min. Eng., vol. 39, 1908, pp. 628–653. Describes experiments made in platinum-foil furnace, in pure, dry air with fluxes, SiO_2, Fe_2O_3, and PbO (separately) with $CaSO_4$. Hydrous $CaSO_4$ loses its combined water at 900° C.; anhydrous $CaSO_4$ is unchanged to 1,200° C., where dissociation begins, with separation of SO_2 and O_2, the rate and degree of dissociation depending on time and temperature. $CaSO_4$ fuses at 1,360° C., being somewhat decomposed. SiO_2 decomposes $CaSO_4$, decomposition beginning at 1,000° C. and finishing at 1,250° C., products being CaO, SiO_2, and gases. It appears that the formation temperature of calcium silicate lies below melting point. Pure Fe_2O_3 begins to decom-

pose $CaSO_4$ at 1,100° C., decomposition ending at 1,250° C. A higher temperature is required than with SiO_2, but the action is more rapid. The decomposition product is $CaO.Fe_2O_3$ (calcium ferrite), which fuses at 1,250° C. This product can dissolve and decompose $CaSO_4$.

78. ———. The behavior of calcium sulphate at elevated temperatures with some fluxes. Trans. Am. Inst. Min. Eng., vol. 40, 1909, pp. 807–808. Authors found that Fe_2O_3 remained unchanged at 1,360° C. in dry air, but at 1,375° C. it lost O_2 (4.4 per cent), became magnetic, and contained some FeO.

79. Hofman, H. O., and Wanjukow, W. The decomposition of metallic sulphates at elevated temperatures in a current of dry air. Trans. Am. Inst. Min. Eng., vol. 43, 1912, pp. 523–577. Heated in current of pure, dry air, $FeSO_4$ fuses to $Fe_2O_3.2SO_3$, which dissociates into Fe_2O_3 and SO_3, noticeably at 492° C., and markedly at 550° to 560° C. $Al_2(SO_4)_3$ begins to dissociate at 590° C., disassociation becoming marked at 639° C. Dissociation of $MgSO_4$ begins at 890° C.

80. Hostetter, J. C., and Sosman, R. B. The dissociation of ferric oxide in air. Jour. Am. Chem. Soc., vol. 38, 1916, pp. 1188–1198. Describes experiments that showed a measureable dissociation of Fe_2O_3 in air at all temperatures between 1,100° and 1,300° C. and indicated that dissociation increases with increase of temperature.

81. Johnson, W. McA. The reducibility of metallic oxides as affected by heat treatment. Trans. Am. Inst. Min. Eng., vol. 47, 1914, pp. 219–231. Discusses H. O. Hofman's experiments to determine comparative reducing effects of CH_4, CO, illuminating gas, and producer gas on hard hematite. The tests showed that, irrespective of the reversibility of the reaction, the progress of the reduction was not simply a function of the time and the temperature.

82. Johnston, John. The thermal dissociation of $CaCo_3$. Jour. Am. Chem. Soc., vol. 32, 1910, pp. 938–946. States that dissociation begins at 587° C. and is complete at 898° C., at atmospheric pressure.

83. Kinnison, C. S. A note on the reduction of Fe_2O_3. Trans. Am. Cer. Soc., vol. 16, 1914, pp. 136–143. The reaction $3Fe_2O_3 \rightleftarrows 2Fe_3O_4 + O_2$ in clays was studied by adding 5 per cent Fe_2O_3 in the form of $Fe_2(OH)_6$ to kaolin and various mixtures of SiO_2 and kaolin and heating the mixture until a black color developed. Author concludes that (1) the smaller the Al_2O_3–SiO_2 ratio the lower the temperature of formation of "ferrous silicate"; (2) at comparatively low temperatures fluxes augment the coloring power of iron; and (3) in mixtures with large amounts of fluxes, "reduction is associated with solution." The dissociation temperature is about 1,210° to 1,220° C. Fluxes contribute to lowering of the transition point.

84. Kremann, R. K. The application of physico-chemical theory to technical processes and manufacturing methods. London, 1913, pp. 115–131. Gives equilibrium data on the systems of Fe and its oxides and of C and its oxides.

85. Mathesius, Ludwig. Untersuchungen über die Reduzierbarkeit von Eisenerzen in strömenden Gasen. Stahl und Eisen, Bd. 34, 1914, pp. 866–872. Describes method for determination of Fe, Fe'', Fe'''. Ores were heated at 600°, 700°, and 900° C. in a current of illuminating gas. Concludes that true equilibrium is not reached in a moving gas stream and that the degree of reduction is largely dependent on the nature of the ore and the exposed surface. Reduction is difficult in a sintered product.

86. Preuner, G. Das Gleichgewicht zwischen Eisen, Eisenoxyduloxyd, Wasserstoff und Wasserdampf. Ztschr. phys. Chem., Bd. 47, 1904, pp. 385–417. Gives equilibrium data for system Fe, Fe_3O_4, H_2, and H_2O. (See pp. 44–46 of this bulletin.)

87. Rhead, T. F. E., and Wheeler, R. V. The effect of temperature on the equilibrium $2CO \rightleftarrows CO_2 + C$. Jour. Chem. Soc., vol. 97, 1910, pp. 2178–2179. Wood charcoal was heated 48 to 240 hours at temperatures of 850° to 1,200° C., and the equilibrium constant K was calculated at the various temperatures from the analysis of the resulting gases.

88. ——— The effect of temperature and of pressure on the equilibrium $2CO \rightleftarrows CO_2 + C$. Jour. Chem. Soc., vol. 99, 1911, pp. 1140–1153. Tables are given showing the composition of the resulting gases at temperatures of 800° to 1,100° C. and pressures of 1.11 to 1.33 atmospheres.

89. ——— The rate of reduction of carbon dioxide by carbon. Jour. Chem. Soc., vol. 101, 1912, pp. 831–845. Critical discussion of results of other investigators. Concludes that in manufacture of producer gas temperatures should be 1,100° C. or more to insure reduction of CO_2 to CO.

90. Schenck, Rudolph. Physikalische Chemie der Metalle. Halle, 1909, 193 pp. Pages 112 to 136 are on oxidation and reduction; pages 137 to 165 treat of blast-furnace gases and equilibrium of various systems, such as CO_2, CO, C, Fe, Fe_2O_3, etc.

91. Sosman, R. B. Some problems of the oxides of iron. Jour. Wash. Acad. Sci., vol. 7, 1917, pp. 55–72. Reviews present knowledge regarding the oxides of iron relative to the phase rule. Discusses the relation of iron oxides to certain geological problems.

92. Sosman, R. B., and Hostetter, J. C. The oxides of iron. 1. Solid solutions in the system Fe_2O_3–Fe_3O_4. Jour. Am. Chem. Soc., vol. 38, 1916, pp. 807–833. Reports investigation undertaken as basis for study of iron-bearing silicates at high temperatures, giving results of measurements of dissociation pressures of the iron oxides made in vacuum furnace.

93. ———. The reduction of iron oxides by platinum, with a note on the susceptibility of iron-bearing platinum. Jour. Wash. Acad. Sci., vol. 5, 1915, pp. 293–303. States that platinum acts on hematite and magnetite at 1,200° C. under low O_2 pressures, absorbing Fe and giving off O_2; reacts in the same way with magnetite at 1,600° C. and O_2 at atmospheric pressure.

94. Stoffel, A. Ueber die Reaktion zwischen Kohlenoxyd und Eisen. Ztschr. anorg. Chem., Bd. 84, 1914, pp. 56–76. Reports results of a study of velocity of formation of $Fe(CO_5)$ from Fe and CO and effect of various factors on the velocity. When CO was passed over Fe powder at a pressure of 1 atmosphere the reaction was at first rapid, becoming slower as the amount of CO absorbed increased, but still continuing after one week. Decomposition occurred above 60° to 70° C. and was complete at 100° C.

95. Walden, P. T. On the dissociation pressure of ferric oxide. Jour. Am. Chem. Soc., vol. 30, 1908, p. 1350. Author determined the pressure curve corresponding to the reaction $3Fe_2O_3 \rightleftarrows 2Fe_3O_4 + O$. At about 1,350° C. the pressure was found to be 160 mm.

96. Wartenberg, H. von. Ueber die Reduktion der Kieselsäure. Ztschr. anorg. Chem., Bd. 79, 1912, pp. 71–87. States that partial pressure of Si vapor, assuming that it is monatomic, is 3×10^{-3} at 1,315° C., and that SiO_2 is reduced and sublimes in H_2 and also in CO at 1,350° to 1,400° C.

MELTING AND SOFTENING TEMPERATURES OF OXIDES, SULPHIDES, SILICATES, MIXTURES, AND SLAGS.

97. Akermann, R. Ueber die Bildungstemperaturen der Hochofenschlacken. Stahl und Eisen, Jahrg. 10, 1890, pp. 424–425; abstracted in Jour. Iron and Steel Inst., 1890, pt. 2, p. 760. Discusses Gredt's determinations of the formation temperatures of slags and compares his results with those obtained by the calorimetric method. The latter method gave lower results. Gives the specific heat of blast-furnace slags as between 0.29 and 0.33.

98. Becker, E. Ueber das Zustandsdiagramm Schwefel-Eisen und den durch Schwefel hervorgurfen Rotbruch. Stahl und Eisen, Bd. 32, 1912, pp. 1017–1021; abstracted in Jour. Iron and Steel Inst., vol. 86, pt. 2, 1912, pp. 604–605. Deals with equilibrium of the system iron-sulphur; gives eutectic mixture as 84.6 per cent iron sulphide and 15.4 per cent iron, melting at 985°.

99. Bishof, Carl. Relative Schmelzbarkeit der Silikate des Eisens, des Kalks, der Magnesia und der Thonerde. Dingl. poly. Jour., Bd. 165, 1862, pp. 378–381. Gives relative fusibilities of iron and lime silicates.

100. Bleininger, A. V., and Brown, G. H. The testing of clay refractories, with special reference to their load-carrying capacities at furnace temperatures. Tech. Paper No. 7, U. S. Bureau of Standards, 1911, 78 pp. Discusses Seger cones and their uses; influence of viscosity; and effects of various constituents of fire clays upon the softening temperatures. *See also* No. 42.

101. Boudouard, O. Experiments on the fusibility of slags. Jour. Iron and Steel Inst., vol. 67, 1905, pp. 339–382. Fletcher gas furnace was used for temperatures up to 1,450° C., and Deville furnace for higher temperatures. Gives softening temperatures of silicates of Al_2O_3 and of CaO; of aluminates of CaO; and of aluminocalcic silicates. Results are represented by triaxial diagrams. Also gives softening temperatures of a number of industrial slags.

102. Cobb, J. W. The synthesis of a glaze, glass, or other complex silicate. Jour. Soc. Chem. Ind., vol. 29, 1910, pp. 69–74, 250–259, 333–336, 399–404, 608–614, 799–802. Experiments were performed in laboratory muffle furnace and in large works furnace. Draws conclusions on progress of slag formation by determining ingredients that were (1) "quickly soluble" (SiO_2, CaO, etc., extracted by N–HCl in one hour), (2) "slowly soluble" (successive treatment with N–HCl, 24-hour periods, until only trace resulted) and (3) "silica soluble in Na_2CO_3" (SiO_2 extracted by hot 5 per cent Na_2CO_3 in 15 minutes). Describes physical condition or degree of progress toward fusion. Gives results with two samples of broken glassware.

103. Day, A. L., and Sosman, R. B. The melting points of minerals in the light of recent investigations on the gas thermometer. Am. Jour. Sci., vol. 31, 1911, pp. 341–349. Gives a series of melting points, with bibliography.

104. Dittler, Emil. The melting points of potassium-sodium feldspars. Tech. Min. Petr. Mittl., Bd. 31, 1914, pp. 513–522; Keram. Rundschau., Bd. 21, 1914, pp. 390–391; Chem. Abs., vol. 8, 1914, p. 222. Presents table showing chemical analysis and mineralogical constitution of feldspars used and their fusion temperatures. Melting points of albite silicates lowered by addition of orthoclase silicates up to 40 to 43 per cent of latter; with larger proportion the melting point is raised.

105. Flach, G. Ueber die Wirkung von Metalloxyden auf den Schmelzpunkte von Quarz-Zettlitzer-Kaolin-Mischungen. Sprechsaal. Jahrg. 44, 1911, pp. 171–173, 187–189, 205–207, 219–220, and 626. Gives tables showing effects of various oxides, among them CaO, MgO, Na_2O, K_2O, FeO, on mixtures of quartz and Zettlitz kaolin. Temperature measurements expressed in terms of Seger cones. Revision of works of Richter, Bishof, Seger, and others.

106. French, A. T. Some analyses of copper blast furnace slags and determinations of their melting points. Mining Sci., vol. 61, 1910, pp. 133–134. Used muffle furnace and Seger cones; did not consider temperature even throughout muffle. Results, for a series of 10 slags, show that slag of highest melting point contained most Al_2O_3 and that slag of lowest melting point contained least Al_2O_3.

107. Fulton, C. H. The constitution and melting points of a series of copper slags. Trans. Am. Inst. Min. Eng., vol. 44, 1912, pp. 751–780. Used a carbon-resistance furnace and, by thermal method, determined melting points of a number of copper slags. Concludes that slags of lowest melting points lie between 1.4 and 1.65, silicate degree.

108. Gredt, Paul. Ueber die Bildungs-temperaturen der Hochofenschlacken. Stahl und Eisen, Bd. 9, 1889, pp. 756–759. Gives formation temperatures of a number of synthetic blast-furnace slags, composed of SiO_2, Al_2O_3, and CaO, and SiO_2, Al_2O_3, CaO, and MgO. Expresses temperatures in terms of Seger cones.

109. Hofman, H. O. Some experiments determining the refractoriness of fire clays. Trans. Am. Inst. Min. Eng., vol. 24, 1894, pp. 42–66. He states that Seger cones are not true pyrometers, as the change from a mixture to a compound is not a function of temperature alone, but depends also on the manner of firing, the time of heating, and the form of furnace.

110. ———. The temperatures at which certain ferrous and calcic silicates are formed in fusion, and the effect upon these temperatures of the presence of certain metallic oxides. Trans. Am. Inst. Min. Eng., vol. 29, 1899, pp. 682–721. Describes softening-temperature determinations made in a gas furnace having a reducing atmosphere, the slag mixture being molded into small cones, $\frac{25}{32}$ by $\frac{3}{8}$ inch, and the temperatures being measured with Seger cones. Gives table showing effect on softening point of replacing SiO_2 with Al_2O_3 (temperature increases with increase of Al_2O_3); also table showing effect of replacing FeO and CaO with Al_2O_3.

111. Hofman, H. O., and Stoughton, B. Does the size of the particle have any influence in determining the resistance of fire clays to heat and to fluxes? Trans. Am. Inst. Min. Eng., vol. 28, 1898, pp. 440–444. Concludes that, as tables show only slight difference in temperature of fusion between pulverized and rough samples, the laboratory method of grinding to a powder will give results directly comparable with those obtained in practice on larger and coarser samples.

112. Howe, H. M. Preliminary note on the thermal properties of slags. Trans. Am. Inst. Min. Eng., vol. 18, 1890, pp. 724–747. Used a gas furnace and calorimetric method of determining temperature. Gives softening temperatures of various CaO, MgO, and SiO_2 mixtures, both with and without previous heating to higher temperature. Gives melting point of $CaO.SiO_2$ as 1,381° C. Thinks that formation temperatures are higher than melting temperatures.

113. ———. Metallurgical laboratory notes. Boston Testing Laboratories, Boston, 1902, pp. 46–69. Discusses melting points of silicates. Uses cone method, making pyramid with ¾-inch base and 1½ inches high, of 100-mesh material, with a binder of 10 to 15 per cent vaseline mixed with heavy lubricating oil.

114. Johnson, W. McA. A new apparatus for determining the melting points of slags. Jour. Chem. Met. Min. Soc. South Africa, vol. 7, 1906–7, p. 98. Slag allowed to drop through hole in cylinder of Acheson graphite.

115. Kanolt, C. W. Melting points of fire bricks. Tech. Paper No. 10, U. S. Bureau of Standards, 1912, 17 pp. Gives melting points of 54 samples of fire bricks, including fire clay, bauxite, silica, magnesia, and chromite brick, determined in Arsem vacuum furnace with Morse optical pyrometer. Melting points were as follows: Kaolin, 1,740° C.; pure Al_2O_3, 2,010° C.; pure SiO_2, 1,750° C.; bauxite, 1,820° C.; bauxite clay, 1,795° C.; and chromite, 2,180° C.

116.———. Melting points of some refractory oxides. United States Bureau of Standards Bull., vol. 10, No. 2, 212, 1914, pp. 295–314. With same methods as described in Technologic Paper 10, author determined following melting points: MgO, 2,800° C.; CaO, 2,572° C.; Al_2O_3, 2,050° C.; Cr_2O_3, 1,990° C.; and Pt. 1,755° C.

117. Kultascheff, N. V. Ueber die Schmelzpunkte von Calciumsilikat ($CaSiO_3$), Natriumsilikat (Na_2SiO_3) und ihren Mischungen. Ztschr. anorg. Chem. Bd. 35, 1903, pp. 187–193. Gives tables of melting points, from pure Na_2SiO_3 at 1,007° C. to mixture of 15 per cent Na_2SiO_3 and 85 per cent $CaSiO_3$ at 1,150° C. (Melting pure $CaSiO_3$ above 1,400° C). Used thermal method in getting freezing-point curve.

118. Leitmeier, H. von. Zur Kenntnis der Schmelzpunkte von Silikaten. Der Einfluss der Korngrösse auf den Schmelzpunkte. Bestimmung des Schmelzpunktes einiger Silikate durch längeres Erhitzen. Ztschr. anorg. Chem. Bd. 81, 1913, pp. 209–232. Used Heraeus platinum furnace, with thermocouple. Gives sintering and melting points for series of natural minerals, showing influence of size of particles. Difference for coarse (3 to 4 mm. diameter) and fine (40 mm. in diameter) particles ranged up to 70° C. Leitmeier's melting temperatures are really softening temperatures, that is, none represent the equilibrium point of solid liquid.

119. Loebe, R., and Becker, E. Das System Eisen-Schwefeleisen. Ztschr. anorg. Chem., Bd. 77, 1912, pp. 301–319. The two components form no mixed crystals, though Fe may be slightly soluble in FeS. Melting point of FeS is 1,198° C.

120. Mathesius, W. Die Entstehung der Schlacken in hüttenmännischen Prozessen; die Konstitution der Schlacken, ihre industrielle Verwertung. Stahl und Eisen, Bd. 24, 1904, pp. 1000–1007. Discusses origin and utilization of slags; does not give formation or melting points or results of original work.

121. Nacken, R. Vergleich der optischen und der thermischen Methode zur Bestimmung von Schmelz-temperaturen. (Compares optical and thermal methods for the determination of melting points.) Centralbl. Min. Geol., 1913, pp. 328–337; Chem. Abs. vol. 7, 1913, p. 3097. Shows discrepancies existing in values given in literature. Author states that by great care he obtained optical results that agree well with those obtained by thermal method.

122. ———. Ueber die Bestimmung der Schmelz-temperaturen von Silikaten auf optischen Wege (determination of fusion temperature of silicates by an optical method). Silikat Ztschr., Jahrg. 1, 1913, pp. 65–67; Chem. Abs., vol. 7, 1913, p. 4053. Substance was heated in platinum-resistance furnace, and observations were made through polarizing microscope; melting point is given as temperature at which interference colors disappear.

123. Rieke, R. Das Richters'sche Gesetz. Ein Beitrag zur Kenntnis der Flussmittelwirkung. Sprechsaal, Jahrg. 43, 1910, pp. 198–199, 214–216, 229–232. Discusses an extensive investigation of the fluxing action of the basic oxides, used in the ceramic industries, on Zettlitz kaolin and mixtures of quartz and kaolin. The raw materials, as kaolin, quartz, limestone, iron oxide, etc., were ground wet in a ball mill. Some of the mixtures were ground dry, ignited in a kiln, and again ground. The softening points of the mixtures were determined in a carbon resistance furnace by comparison with standard pyrometric cones.

124. ———. Ueber die Schmelz-barkeit von Kalk-Tonerde-Kieselsäure Mischungen. Stahl und Eisen, Bd. 28, 1908, pp. 16–19; Abs. Jour. Soc. Chem. Ind., vol. 27, 1908, p. 123. Discusses fusibility of lime-alumina-silica mixtures. The mixture was molded into small cones, and the softening points were measured by Seger cones. Mixtures with more than 50 per cent CaO crumbled into powder when cooled slowly.

125. Ruff, Otto. Arbeiten im Gebiet hoher Temperaturen: Ueber das Schmelzen und Verdampfen unserer feuerbeständigsten Oxyde im elektrischen Vakuumofen. Ztschr. anorg. Chem., Bd. 82, 1913, pp. 373–400. Further work on material mentioned in No. 126 following.

126. ———. Ueber einen elektrischen Vakuumofen. Ber. Deut. Chem. Gesell., Bd. 43, 1910, pp. 1564–1574. Describes carbon-tube-resistance vacuum furnace and its uses in melting and vaporizing oxides. Gives melting points, as follows: Pt, 1,745° to 1,755° C.; kaolin, 1,912° C.; and Al_2O_3 (in N_2), 2,065° C.

127. Ruff, O., and Goeke, O. Ueber das Schmelzen und Verdampfen unserer sogenannten hochfeuerfesten Stoffe. Ztschr. angew. Chem., Bd. 24, 1911, pp. 1459–1465. Discusses investigations on the melting and vaporization of highly refractory materials. An electric vacuum furnace and a Wanner pyrometer were used. Gives melting point of Al_2O_3 at 2,020° C.

128. Seger, H. A. Collected writings; translated by members of the American Ceramic Society and edited by A. V. Bleininger. Easton, Pa., 1902, 2 vols., 1157 pp. Pages 519 to 520 are on "The effect of titanic acid on the fusibility of fire clays"; author considers that TiO_2 has greater fluxing power than SiO_2. Pages 1005 to 1027 give data on use of Seger cones.

129. Smolensky, S. Schmelzversuche mit Bisilikaten und Titanaten. Ztschr. anorg. Chem., Bd. 73, 1911–12, pp. 293–303. States that thermal method was used to determine melting points of various mixtures of $CaSiO_3$ and $CaTiO_3$, $BaSiO_3$, and $BaTiO_3$, and $MnSiO_3$ and $MnTiO_3$. A kryptol resistance furnace and a Pt–PtRh thermocouple were used. In general, the addition of TiO_2 lowered the melting point.

130. Sprechsaal Kalender. Schmelzpunkte. Jahrg. 8, Coburg, 1916. Gives melting points of oxides, silicates, and various mixtures.

131. Steffe, Herman. Ueber die Bildungstemperaturen einiger Eisenoxydul-Kalkschlacken und einiger kalkfreien Eisenoxydul-schlacken, deren Kenntniss für das Verschmelzen der Bleierze Bedeutung hat. Dissertation. Berlin, 1906. Gives formation temperatures of some ferro-calcic and ferrous slags. Observations made of sintering temperature of mixture when heated in a platinum boat in a current of nitrogen. Temperatures were measured with thermocouple.

132. Stein, G. Ueber die Darstellung einiger Silikate. Ztschr. anorg. Chem., Bd. 55, 1907, pp. 159–174. The optical method, with Wanner pyrometer, was used for determining melting and inversion points of various silicates (Al_2O_3, FeO, etc.). Discusses formation of Ca, Sr, and Ba carbides in the presence of SiO_2.

133. Treitschke, W., and Tammann, G. Ueber das Zustands-diagramm von Eisen und Schwefel. Ztschr. anorg. Chem., Bd. 49, 1906, pp. 320–335. Determines cooling curve by thermal method, obtaining two "stop points"—one at 970° C. and one at 1,400° C.

134. White, W. P. Melting-point determinations. Am. Jour. Sci., vol. 28, 1909, pp. 453–473. Gives theoretical and mathematical considerations regarding the thermal method for determining melting points. Cites reasons for the variations that have been observed and mentions precautions.

135. ———. Melting-point methods at high temperatures. Am. Jour. Sci., vol. 28, 1909, pp. 474–489. Describes furnace and thermo elements used, giving best methods for making melting-point determinations up to 1,600°.

136. Anonymous. Formation temperatures of ferrous oxide-lime-silica slags. Trans. Am. Inst. Min. Eng., vol. 31, 1901, p. 862; curves plotted from Hofman's paper. Trans. Am. Inst. Min. Eng., vol. 29, 1899, pp. 682–721, showing effect on fusion temperature of replacing CaO by MgO and slag constituents by Al_2O_3. *See also* No. 63.

137. ———. Schmelzpunkte von Oxygen, Silikaten, Boraten und Aluminaten und deren eutektischen Gemischen. Sprechsaal, Jahrg., 44, 1911, pp. 710–713, 729–731, and 744–746. Presents, from various sources, a tabulation of results conveniently arranged.

PYROMETRIC CONES; THEIR USE IN MEASURING TEMPERATURE.

138. American Ceramic Society. Report of committee on Seger cones. Trans., vol. 3, 1901, pp. 180–192. The committee was appointed to investigate reasons for unsatisfactory behavior of low-melting cones, mainly those containing Fe_2O_3. Suggests that boric acid, affected by S in the gas, and perhaps $CaCo_3$, changed to $CaSO_4$ by S in gas may be the cause of trouble.

139. Ashley, H. F. A comparison of the various makes of Seger cones on the market. Trans. Am. Cer. Soc., vol. 8, 1906, pp. 159–167. Compares German, English, and various American makes of cones in 48-hour burns in ceramic kilns.

140. Brown, G. H., and Murray, G. A. The function of time in the vitrification of clays. Tech. Paper No. 17, U. S. Bureau of Standards, 1913, 26 pp. Shows that rate of heating has decided bearing on the softening points of Seger cones and, in general, that the effect of time in firing of clay products is important.

141. Geigsbeck, S. The fusing points of Seger cones expressed in degrees. Trans. Am. Cer. Soc., vol. 6, 1904. pp. 94–108. Discusses the advisability of expressing the melting points of cones in degrees. Orton and others opposed to such procedure.

142. ———. The melting points of pyrometric cones under various conditions. Trans. Am. Cer. Soc., vol. 14, 1912, pp. 849–871. Concludes that rate of heating has much to do with melting points of Seger cones and that they can not be used for measuring temperature.

143. Gorton E. E. Remarks on Seger cones. Trans. Am. Cer. Soc., vol. 2, 1900, pp. 60–73. General discussion of application of Seger cones to ceramic industry, with especial reference to variable indications of the low-melting cones containing Fe_2O_3.

144. Hoffman, Dr. G. Prüfung der Segerkegel. Tonind. Ztg., Bd. 33, 1909, pp. 1577–1580. Measured softening points of cones 26 to 35 in an iridium tube furnace using Ir-Ir-Ru thermocouple. His results are lower than estimated melting temperatures by 5° to 60° C.

145. ———. Prüfung der Segerkegel. Tonid. Ztg., Bd. 35, 1911, pp. 1099–1100; Sprechsal, Bd. 44, 1911, pp. 143–145. Continuation of work described in 144, in which author compares softening temperatures as obtained in iridium furnace with optical pyrometer or thermocouple, in the laboratory of the Reichsanstalt, with average values obtained in industrial ceramic kilns. The fusion temperatures are 35° to 95° C. lower in ceramic kilns than the accepted Seger cone figures, and 40° to 110° C. lower than results obtained in iridium furnace.

146. Rieke, R. Die Schmelzpunkte der Segerkegel 022–15: Sprechsaal, Bd. 44, 1911, pp. 726–729, 741–744; Tonind. Ztg., Bd. 35, 1911, pp. 1751–1757. Although some cones fused at a temperature lower than accepted figures, on slow heating other cones showed opposite behavior. All cones fused at a lower temperature when heated slowly than with rapid heating; differences were as great as 60° to 100° C.

147. Simonis, M. Segerkegel 022 bis 7. Tonind. Ztg., Bd. 32, 1908, pp. 1764–1768. The object of the investigation described in this article was to overcome troubles encountered with cones that fuse at low temperatures. Cones tested were free from lead.

148. Sosman, R. B. The physical chemistry of Seger cones. Trans. Am. Cer. Soc., vol. 15, 1913, pp. 482–498. Shows the influence of eutectics on the deformation temperatures of Seger cones. States that the existance and influence of these eutectics makes temperature measurements by cones uncertain.

149. Zimmer, W. H. Practical experience with pyrometers. Trans. Am. Cer. Soc., vol. 1, 1899, pp. 22–42. Seger cones measure heat effect rather than temperature, therefore are of especial value in ceramic industries in firing kilns.

150. *See also* No. 128.

PYROMETERS AND THE MEASUREMENT OF HIGH TEMPERATURE.

151. Bidwell, C. C. Actual and black-body temperatures. Sci. Abs. vol. 17, A, 1914, p. 539; Phys. Review, vol. 3, 1914, pp. 439–449. Comparisons are made between actual and black-body temperatures of Fe, Au, Ag, Cu, and Ni, both solid and liquid, over the range 1,000° to 2,000° C., absolute.

152. Burgess, G. K. The estimation of the temperature of copper by optical pyrometers. U. S. Bureau of Standards Bull., vol. 6, No. 1, October, 1909, pp. 111–119. Describes experiments in which the temperature

of solid and liquid copper was measured with thermocouple and various optical pyrometers.

153. Burgess, G. K., and Le Chatelier, H. The measurement of high temperatures. Third ed., 1912, 510 pp. Authoritative discussion on the thermo-electric pyrometer, the optical pyrometer, and the standardization of pyrometers. The discussion covers thoroughly the pyrometers used in the work described in this bulletin, giving the theory of and directions for their use.

154. Burgess, G. K., and Wartenberg, R. G. The emissivity of metals and oxides; measurements with the micropyrometer. Jour. Franklin Inst., vol. 178, 1914, p. 778. Describes a micropyrometer of the Holborn-Kurlbaum type.

155. Foote, P. D. A note on the calibration of optical pyrometers. Met. Chem. Eng., vol. 11, 1913, pp. 97–98. Describes an inexpensive graphite crucible for getting black-body temperatures—a substitute for the rather expensive Lummer-Kurlbaum apparatus. Author determined melting and freezing points of Cu, Ag-Cu eutectic, and Sb.

156.———. Cold-junction corrections for thermocouples. Met. Chem. Eng., vol. 11, 1913, pp. 329–333. Corrections and mathematical calculations for observations in which cold junction is not kept at zero. Describes various devices for eliminating cold-junction corrections.

157. Harker, J. A. Measurements of elevated temperatures. Céramique, t. 16, 1913, p. 40; Chem. Abs., vol. 7, 1913, p. 1961. Presents four tables giving temperatures of a few sources of heat, practical capacity of instruments for measurement of temperature, and correspondence of colors and temperatures.

158. Northrup, E. F. Temperatures and the properties of matter. Met. Chem. Eng., vol. 10, 1912, pp. 352–356. Reviews recent advances in the establishment of a standard scale of temperatures and discusses properties of matter at very high and very low temperatures.

159. Pirani, M. von. Optical temperature measurements. Ber. physik. Gesell., 1911, pp. 19–25; Chem. Abs., vol. 5, 1911, p. 1553. Describes a proposed convenient method for correcting temperature measurements with optical pyrometers.

160. Pirani, M. von, and Meyer, A. R. Über die Eichung von Pyrometerlampen vermittels zwei Temperaturfixpunkte. (The calibration of pyrometer lamps by means of two fixed temperatures.) Ztschr. wiss. Photochem., Bd. 10, 1912, pp. 135–137; Chem. Abs., vol. 6, 1912, p. 1251. For filiments, such as are used in Holborn-Kurlbaum pyrometer.

161. Waidner, C. W., and Burgess, G. K. Platinum resistance thermometry at high temperatures. U. S. Bureau of Standards Bull. 2, vol. 6, November, 1909, pp. 149–230. Discusses the Pt–PtRh thermocouple, giving determinations of a number of melting points, with bibliography.

162. Whipple, R. S. Modern methods of measuring temperature. Proc. Inst. Mech. Eng., vol. 77, 1913, pp. 717–775. Gives history of pyrometry and discusses various means of measuring temperatures, with special reference to the use of pyrometers in industrial plants.

LABORATORY FURNACES FOR HIGH-TEMPERATURE WORK.

163. Arsem, W. C. The electric vacuum furnace. Trans. Am. Electrochem, Soc., vol. 9, 1906, pp. 153–171. Describes furnace with heating unit made from graphite in form of a helix, all inclosed in gas-tight, water-cooled metal jacket. Author obtained temperatures up to (perhaps beyond) 3,000° C., using maximum of 13 k. v. a.

164. Clerc, L., and Minet, A. Sur un nouveau four électrique à arc, applicable aux recherches de laboratoire. Compt. rend., t. 146, 1908, p. 227; Moniteur Sci., t. 22, 1908, p. 154. Describes an electric-arc furnace of 1 to 2 kilowatt capacity, manufactured in Paris.

165. ———. Untersuchungen über den electrischen Öfen. Electrochem. Ztschr., Bd. 15., 1908, pp. 97–100; Chem. Abs., vol. 2, 1908, p. 3196. Describes an arc furnace, Moisson type, with values for current, voltage, etc.

166. Colhane, D. F., and Bard, E. E. An efficient electric furnace for high temperatures. Met. and Chem. Eng., vol. 10, 1912, pp. 461–463. A carbon-resistance furnace, external parts of cement, refractories of alundum. Current consumption, $\frac{1}{2}$ to 1 kilowatt.

167. Day, A. L., and Allen, E. T. Temperature measurements up to 1,600° C. Phys. Review, vol. 19, 1904, pp. 177–186. Describes platinum-wire (internal winding) vertical furnace for melting-point determinations.

168. Field, A. L. An electrical-contact vapor-pressure thermoregulator. Jour. Am. Chem. Soc., vol. 36, 1914, pp. 72–76. Describes U-tube, saturated-vapor type of thermoregulator, which is used in connection with molybdenum-wire furnace described in this bulletin on pp. 46 to 49.

169. Frederich, K. Über einiger neuere Formen von elektrisch geheizten Laboratoriumsöfen (several new forms of laboratory furnaces). Metallurgie, Jahrg. 4, 1907, pp. 778–781; Chem. Abs., vol. 2, 1908, p. 1084. Describes platinum-foil and kryptol furnaces for high-temperature work.

170. Goecke, Otto. Die elektrische Vakuumöfen und seine Verwendung (the electric vacuum furnace and its use). Metallurgie, Jahrg. 8, 1911, pp. 667–676; Chem. Abs. vol. 6, 1912, p. 1401. Carbon cylinder used as heating element.

171. Goodwin, J. H. Granular carbon resistance furnace. Met. and Chem. Eng. vol. 9, 1911, p. 188. Describes furnace $17\frac{1}{2}$ by $12\frac{3}{8}$ by $10\frac{5}{8}$ inches, with heating space 7 by 5 by 3 inches, and shell made of No. 14 sheet steel.

172. Heraeus, W. C. Ueber elektrische Laboratoriumsöfen. Ztschr. Elektrochem., Bd. 18, 1912, p. 143. Comparison of efficiency of Pt and Ni wound furnaces.

173. Hoffman, G. Prüfung der Segerkegel. Sprechsaal, Bd. 44, 1911, pp. 143–145. Describes an iridium tube furnace.

174. Jeffries, Zay. Notes on the granular electric furnace. Met. and Chem. Eng., vol. 12, 1914, pp. 154–157. Describes a convenient type of carbon-resistance furnace, with a resistor of kryptol, granular carbon or graphite, and alundum refractories. Approximately 3 kilowatts is required to attain temperature of 1,600° C.

175. Le Blanc, M. Widerstandofen mit elektrisch geheiztem Nickeldraht. Ztschr. Elektrochem., Bd. 15, 1909, pp. 683–687. Describes furnace of 2-mm. nickel wire wound on Berlin porcelain, used up to 1,300° C. maximum.

176. Malm, W. R. A high-temperature experimental furnace. Met. and Chem. Eng., vol. 13, 1915, p. 70. Describes a crucible-type arc furnace especially adapted for fusion of ferroalloys. Takes 30 to 50 amperes, giving $\frac{1}{2}$-inch arc.

177. Northrup, E. F. A new high-temperature electric furnace. Met. and Chem. Eng., vol. 12, 1914, pp. 31–33. Describes a vertical type furnace, with Acheson graphite heating element, the whole inclosed in Monel-

metal case, with a heating space 12 inches long by 1⅞ inches in diameter. Maximum capacity, 170 amperes at 23 volts. Used for study of properties of metals at high temperatures.

178. ———. Cascade attachment for graphite furnace. Met. Chem. Eng., vol. 12, 1914, pp. 305–306. Discusses a supplementary device for furnace described in No. 177 to raise temperature to melting point of tungsten or higher. Maximum current consumption of 9.5 kilowatts.

179. Pip, W. Zwei neue Laboratoriumsöfen für hohe Temperaturen. Ztschr. Elektrochem., Bd., 16, 1910, pp. 664–667. Describes an oven constructed of MgO and heated by gas, and an oven of the carbon-resistance type.

180. Slade, R. E. An electric furnace for experiments in a vacuum at temperatures up to 1,500° C. Electrician, vol. 70, 1912, p. 293; vol. 71, 1913, p. 479. Describes a furnace consisting of Pt tube 17cm. long, 2 cm. in diameter, and 1 mm. thick, with the ends fitted into water-cooled brass terminals.

181. Sosman, R. B., and Hostetter, J. C. A vacuum furnace for the measurement of small dissociation pressures. Jour. Wash. Acad. Sci., vol. 5, 1915, pp. 277–285. Describes a furnace similar in principal to Slade's furnace, but with alloy tube 80 per cent Pt and 20 per cent Rh as resistor.

182. Tucker. S. A. Electric tube furnace for temperature measurements. Trans. Am. Electrochem. Soc., vol. 11, 1907, pp. 303–306. Describes a carbon-tube furnace with resistor 540 mm. long; part in center turned down thin to increase resistance; about 1 inch in inside diameter; ⅝-inch walls in thick part. Tube surrounded by petroleum coke; whole incased in brick; water-cooled brass electrodes soldered on tube. About 15 volts required to attain a temperature of 3,000° C. A maximum current of 850 amperes is required.

183. ———. A granular-carbon resistance furnace. Trans. Am. Electrochem. Soc., vol. 11, 1907, pp. 307–315. Furnace described makes use of Acheson graphite electrodes. Crushed gas coke for resistor and graphite crucible. Maximum capacity 30 to 35 kilowatts at low voltage—30 to 40 volts—and high current density—maximum 700 amperes.

184. ———. Platinum resistance furnace for melting points and combustions. Jour. Am. Chem. Soc., vol. 29, 1907, pp. 1442–1444. Register is Pt. tape wound on quartz tube, used only for temperatures up to 1,200° C.

185. Tucker, S. A., and Moody, H. R. Improved electric furnace for laboratory use. Jour. Am. Chem. Soc., vol. 23, 1901, pp 473–476. Describes an electric-arc furnace, Moisson type, with capacity of 8½ to 12½ kilowatts at 70 to 90 volts. Either water-jacketed or copper-plated electrodes.

186. Ubbelohde, L. Electrische Laboratoriumsöfen mit Wicklung aus unedlen Metalle. Ztschr. Electrochemie, Bd. 17, 1911, pp. 1002–1003. Compares Ni and Pt wound furnaces for low-temperature work.

187. Winn, R., and Dantsizen, C. Small electric furnace with heating element of ductile tungsten or ductile molybdenum. Jour. Ind. and Eng. Chem., vol. 3, 1911, pp. 770–771. Discusses construction of two types—vertical, crucible type, and horizontal, tube type, each with resistor of ductile tungsten or molybdenum wire protected from oxidation by an atmosphere of H_2

See also Nos. 13, 46, 62, 126.

PUBLICATIONS ON THE UTILIZATION OF COAL AND LIGNITE.

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BULLETIN 27. Tests of coal and briquets as fuel for house-heating boilers, by D. T. Randall. 44 pp., 3 pls., 2 figs. 10 cents.

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